UNIV. OF
TORONTO
LIBRARY

DEPARTMENT OF MINING ENGINEERING
[illegible] 1141
[illegible]
[illegible] N
[illegible] 2
[illegible]

~~Section 52~~
~~No 550~~

Digitized by the Internet Archive
in 2011 with funding from
University of Toronto

http://www.archive.org/details/journalsouthafri09sout

~~Tech~~
C

11531

THE JOURNAL

OF THE

CHEMICAL, METALLURGICAL & MINING SOCIETY OF SOUTH AFRICA.

VOLUME IX.

JULY, 1908 — JUNE, 1909.

101778
17/5/10.

JOHANNESBURG:
Published by the Society: 9, Transvaal University College Building, Eloff Street.

TN
1
S66
v. 9

OFFICERS AND COUNCIL OF THE SOCIETY

JULY, 1908—JUNE, 1909.

Hon. President:

H. E. The Earl of Selborne, K.G., P.C., G.C.M.G.

Hon. Vice-Presidents:

Lionel Phillips (President, Transvaal Chamber of Mines).

Geo. Albu,
F. D. P. Chaplin, M.L.A.,
F. Eckstein,
Sir George Farrar, Kt., D.S.O., M.L.A.,
Sir J. P. FitzPatrick, Kt., M.L.A.,
J. N. de Jongh,
Hennen Jennings,
L. Reyersbach,
Harold F. Strange,
Hon. J. de Villiers, M.L.A., Minister of Mines

Past Presidents:

Andrew F. Crosse	1895—1896.
W. R. Feldtmann	1896—1897.
Charles Butters	1897—1898.
Dr. J. Loevy	1898—1899
John R. Williams	1899—1903.
S. H. Pearce	1903—1904.
W. A. Caldecott	1904—1905.
William Cullen	1905—1906.
Edward H. Johnson	1906—1907.
John Yates	1907—1908.

COUNCIL FOR THE YEAR ENDING JUNE, 1909.

President:

R. G. Bevington.

Vice-Presidents:

T. Lane Carter.
A. McA. Johnston.
James Moir.

Ordinary Members of Council:

E. H. Croghan,
W. R. Dowling,
K. L. Graham,
C. B. Kingston,
A. Richardson,
C. B. Saner,
G. O. Smart,
G. H. Stanley,
J E. Thomas,
A. Whitby,
H. A. White,
J. A. Wilkinson.

With the Past Presidents, who are Members of the Council *ex-officio*.

Hon. Treasurer:

James Littlejohn.

Hon. Auditor:

Alex. Aiken, F.S.A.A.

Hon. Legal Advisers:

Messrs. van Hulsteyn, Feltham, and Fry.

Secretary:

Fred. Rowland, Fellow, Chartered Inst. Secretaries.

Editorial Committee:

R. G. Bevington.
G. H. Stanley.
A. Whitby.
John Yates.

Publications Committee:

E. H. Croghan,
W. R. Dowling,
K. L. Graham,
A. McA. Johnston,
C. B. Kingston,
A. Richardson,
J. A. Wilkinson,
G. H. Stanley (*Convener*).

Finance Committee

J. Littlejohn.
James Moir.
C. B. Saner,
J. A. Wilkinson.

CONTENTS OF VOLUME IX.

PAGE

Officers and Council of the Society ... ... iii.

Papers Read Before the Society ... ... vii.

List of Illustrations ... ... viii.

Journal No. 1.—July, 1908 1

" " 2.—August " 37

" " 3.—September " 69

" " 4.—October " 105

" " 5.—November " 141

" " 6.—December " 181

" " 7.—January 1909 221

" " 8.—February " 253

" " 9.—March " 289

" " 10.—April " 325

" " 11.—May " 373

" " 12.—June " 417

Index to *Journal* 453

PAPERS READ BEFORE THE SOCIETY.

PAGE
Inaugural Address, by R. G. Bevington 2
Amalgamating Plates, The Silver Coating of, by W. A. Caldecott 142
Amalgamation, The Effect on, of Different Intervals of Time between the Dressing of Plates, by G. O. Smart 425
Assaying, Routine, on a Westralian Mine, by W. B. Blyth 184
Atomic Theory, Suggestions for a New, by Dr. J. Moir 334
Battery and Cyanide Gold Smelting, by A. Thomas 6
Blasting with High Explosives, The Theory of, by E. M. Weston 110
Carbon Monoxide in Mine Gas, The Estimation of, by Dr. E. H. Weiskopf 258
Coals, South African, and their Economics, by A. J. Andrews 330
Extractor Boxes, Covers for, by K. L. Graham 325
Gases Resulting from the Use of High Explosives, by W. Cullen 144
Gold Crystals, by A. McA. Johnston 182
Gold-Lead Bullion, A Few Notes on Cupelling, by G. Melvill 157
Gold Precipitates and Bullion, The Smelting of, With Oil Fuel, by A. Yates 429
Gold Produced on the Rand, The Distribution of the, by Dr. T. Kirke Rose 265
Grading Assays and Grinding Efficiencies, by A. Yates 187
Ore Treatment at the Giant Mine, Hartley District, Rhodesia, Description of, by R. C. H. Cooke 152
Patio Process, The, by C. Perez Duarte 105
Pilgrims Rest Gold Fields and Mining Methods, The, by J. Moyle-Phillips 293
Platinum Parting Apparatus, A Cheap, by Prof. G. H. Stanley 256
Potassium, Improved Glasses for Testing for, by Dr. J. Moir 292
Precipitation from Cyanide Solutions by Zinc Shavings and Dust: A Comparison of Results and Costs, by Allen J. Clark 222
Precipitation, Notes on, by Mather Smith 300
Precipitation of Gold from Solution by Carbonaceous Matter, The, by W. A. Caldecott ... 327
Quick-Lime, The Influence of Moist Air on, by James Gray 396
Reminiscences of the Early Rand, by M. H. Coombe 38
Safety Fuse, Testing of, by X Rays, by James Thomas 183
Sands Collecting Vat, Counterweighted Blind for, by C. W. Dowsett 326
Sands Return Contrivance, A Low Fall, by C. W. Dowsett 326
Shaft Sinking, Note on a Problem during, by C. B. Saner 70
Small Mines of Rhodesia, by B. I. Collings 76
Stope Measurements, by O. S. Tonnesen 375
Telluride Gold Ores of Cripple Creek (Colorado), Researches upon the, by the "Portland Metallurgical Society" (Edited by T. B. Crowe) 398
White Labour in Mining, by Tom Johnson 224
Zinc Dust Precipitation at Cerro Prieto, by Robert Linton 74

LIST OF ILLUSTRATIONS.

PAGE
Diagrams illustrating C. O. SCHMITT's discussion on "Notes on Rand Mining" 17
Diagrams illustrating "Repairing a Mortar Box Pile," by A. RICHARDSON ... 25, 26
Diagrams illustrating Reply to Discussion on "Rescue Appliances: Lessons from Glencoe," by H. KESTNER 43
Diagrams illustrating "Economics of Tube Milling," by H. W. FOX 59, 60, 61
Diagrams illustrating "Note on a Problem during Shaft Sinking," by C. B. SANER 70, 71, 72, 73
Diagrams illustrating Reply to Discussion on "Notes on Rand Mining," by TOM JOHNSON, 86, 87, 90, 91, 93
Diagrams illustrating "The Theory of Blasting with High Explosives," by E. M. WESTON, 111, 112, 113, 117
Diagrams illustrating "On Certain Errors in Computing Ore Values," by H. H. KNOX ... 135
Diagrams illustrating "Safety Sinking Hooks," by H. LOUIS 136
Plates illustrating "The Silver Coating of Amalgamating Plates," by W. A. CALDECOTT ... 143
Diagram illustrating "Description of Ore Treatment at Giant Mine," by R. C. H. COOKE ... 155
Plates illustrating "Gold Crystals," by A. McA. JOHNSTON 182
Plate illustrating "Testing of Safety Fuse by X Rays," by JAS. THOMAS 183
Plate and Diagrams illustrating H. MUSSON THOMAS' Discussion on "The Theory of Blasting with High Explosives" 197, 198, 199, 200, 201
Plates illustrating Visit to Premier Diamond Mine 210, 211
Plate and Diagram illustrating G. A. DARLING's Discussion on "Reminiscences of the Early Rand" 228
Diagrams illustrating "A Safety Hook for Shaft Sinking," by T. H. PRISK 246
Plates and Diagrams illustrating "A Cheap Platinum Parting Apparatus," by Prof. G. H. STANLEY 256, 257
Diagram illustrating "The Estimation of Carbon Monoxide in Mine Gas," by Dr. E. H. WEISKOPF 263
Diagrams and Sketch (inset) illustrating "The Pilgrims Rest Gold Fields and Mining Methods," by J. MOYLE-PHILLIPS 294, 295, 296, 298
Diagrams illustrating Reply to Discussion on "Note on a Problem During Shaft Sinking," by C. B. SANER 303, 304
Diagrams illustrating "Cover for Extractor Boxes," by K. L. GRAHAM 326
Diagram illustrating "A Low Fall Sands Return Contrivance," by C. W. DOWSETT ... 326
Diagram illustrating "Counterweighted Blind for Sands Collecting Vat," by C. W. DOWSETT .. 327
Diagrams illustrating "Suggestions for a New Atomic Theory," by Dr. J. MOIR ... 337, 339
Diagrams illustrating "Stope Measurements," by O. S. TONNESEN, 377, 378, 379, 380, 381, 382, 383, 384, 385, 386, 387
Diagram illustrating P. R. ROUX's Discussion on "Suggestions for a New Atomic Theory," ... 393
Diagram illustrating W. PRICE GRIFFITH's Discussion on "The Pilgrims Rest Gold Fields and Mining Methods" 396
Plates illustrating Visit to Simmer Deep 405, 407
Diagram illustrating "The Effect on Amalgamation, of Different Intervals of Time between the Dressings of Plates," by G. O. SMART 426
Diagram illustrating "The Economics of Amalgamation," by J. H. HAYNES 439
Diagram illustrating "Mine Labour Chart," by J. MACDONALD 445

SUPPLEMENT TO THE JOURNAL

OF THE

CHEMICAL, METALLURGICAL & MINING SOCIETY

OF SOUTH AFRICA.

JULY, 1909.

PLATES ILLUSTRATING

THE

INAUGURAL ADDRESS

ON

THE CRYSTALLISATION OF IRON AND STEEL

BY

A. McARTHUR JOHNSTON, M.A., M.I.M.M., F.C.S.

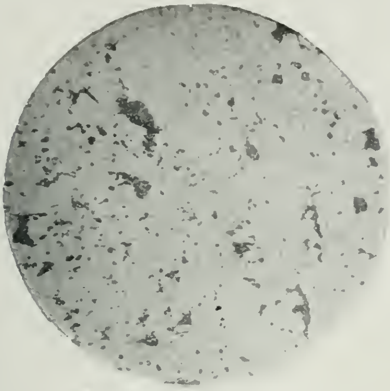


FIG. 1.—Wrought iron, × 100 diameters. Cross section.

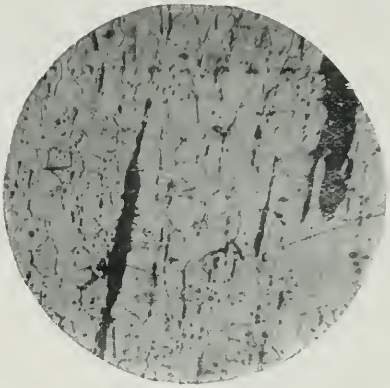


FIG. 2.—Wrought iron, × 100 diameters. Longitudinal section.

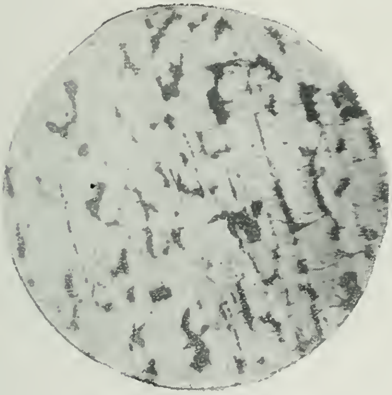


FIG. 3.—Mild steel, × 100 diameter. 0·2% carbon.

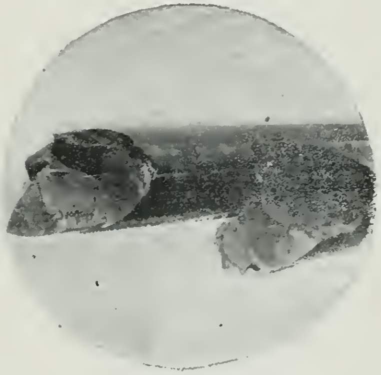


FIG. 4.—Broken cam shaft.

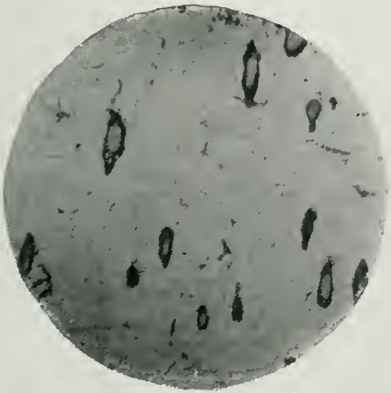


FIG. 5.—Part of cam shaft, × 100 diameter. Wrought iron. Cross section,

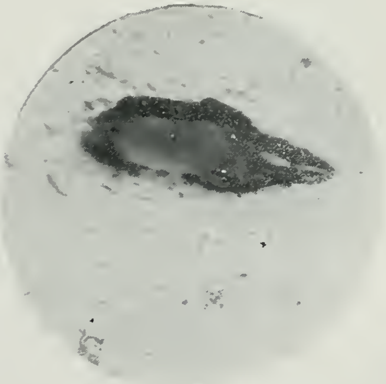


FIG. 6.—Same as Fig. 5, × 400 diameter,

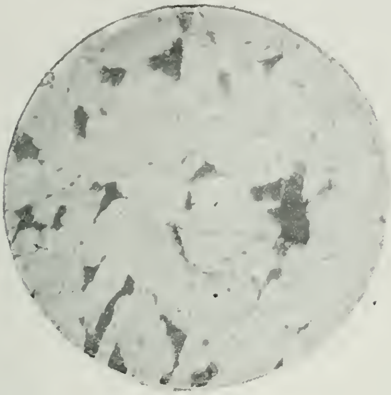


FIG. 7.—Part of cam shaft, × 100 diameter.
Mild steel. Cross section.

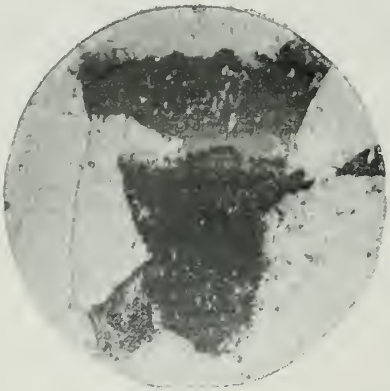


FIG. 8.—Same as Fig. 7, × 400 diameter.

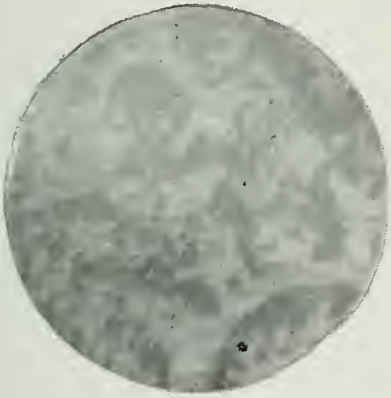


FIG. 9.—Low carbon steel, × 100 diameter.
0·30% carbon.

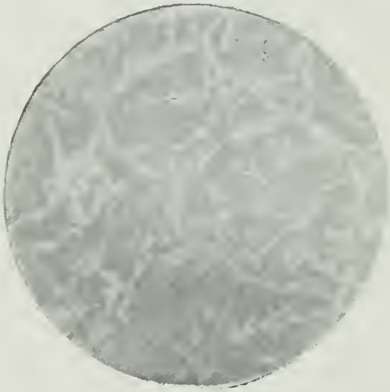


FIG. 10.—Medium carbon steel, × 100
diameter. 0·50% Carbon.

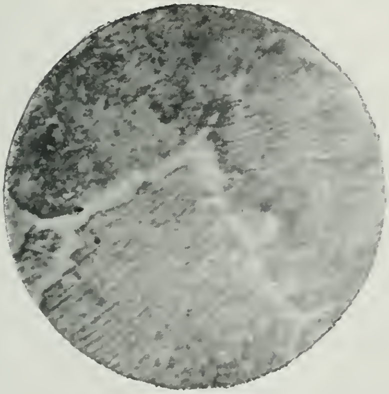


FIG. 11.—High carbon steel. × 1000
diameter. 0·75% carbon.

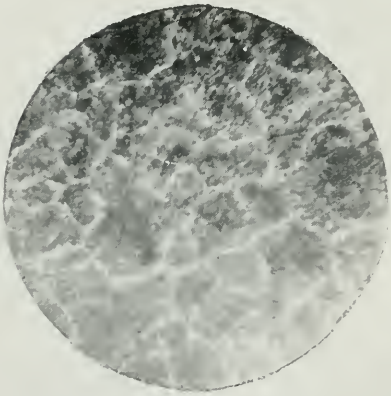


FIG. 12.—High carbon steel, × 100
diameter. 1·25% carbon.

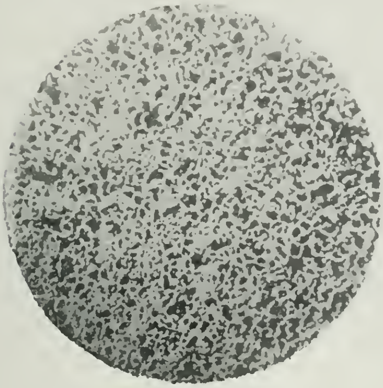


FIG. 13.—Steel.
Grain size at 900°C (Campbell).

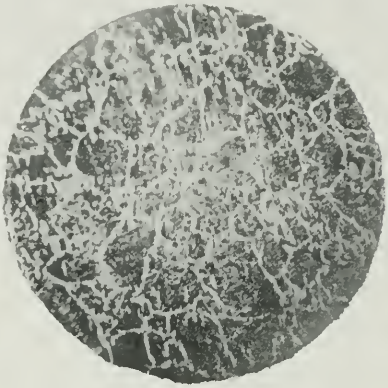


FIG. 14.—Steel.
Grain size at 1200 C (Campbell).

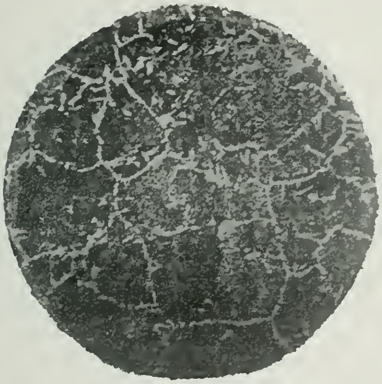


FIG. 15.—Steel.
Grain size at 1300 C (Campbell).

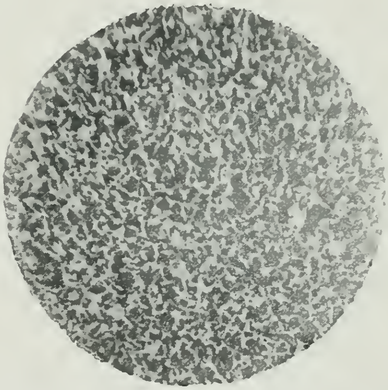


FIG. 16.—Steel (forged). Heated to 800 C
and cooled in air. × 100 diameters.

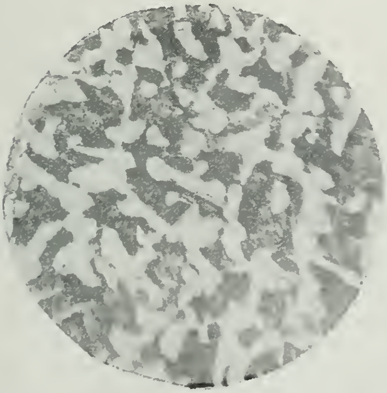


FIG. 17.—Steel (forged). Heated to 1000 C
and cooled in air. × 100 diameter.

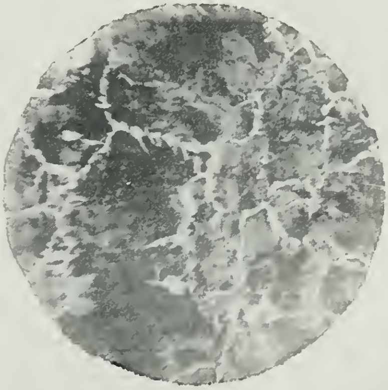


FIG. 18.—Steel (forged). Heated to 1000 C
and cooled in furnace. × 100 diameter.

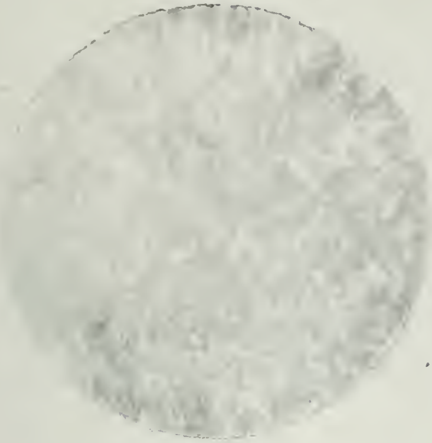


FIG. 19.—Wire, × 100 diameter.
Regular structure.

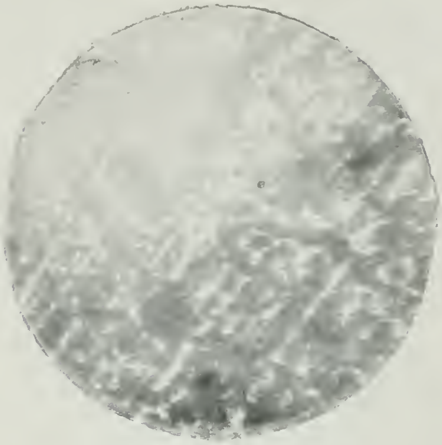


FIG. 20.—Wire, × 100 diameter.
Longitudinal section, showing cold work.

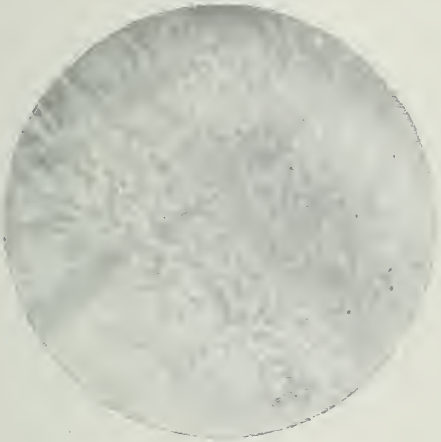


FIG. 21.—Wire, × 100 diameter.
Longitudinal section. Distorted grains and cracks.

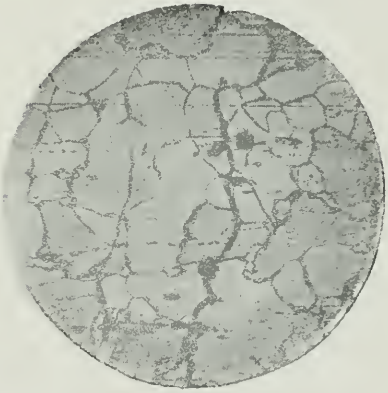


FIG. 22. Polished surface with large crack (Ewring & Humphrey).

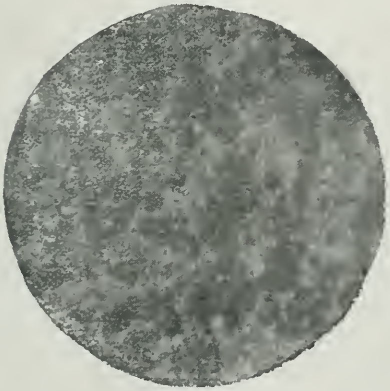


FIG. 23.—Outside section, × 400 diameter
Before use

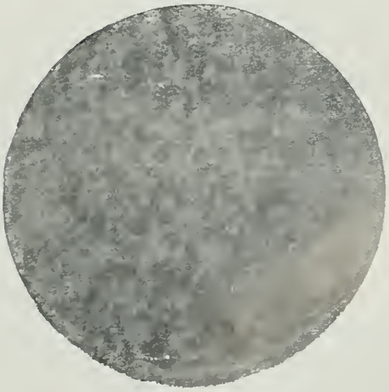


FIG. 24.—Middle section, × 400 diameter.
Before use.

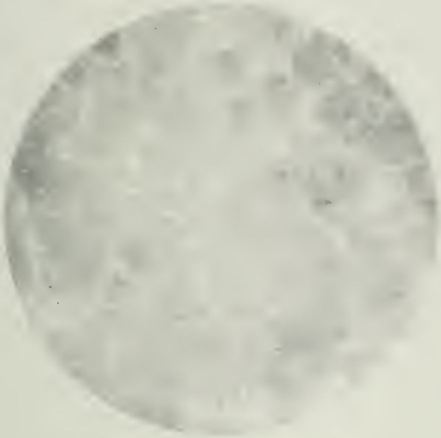


FIG. 25.—Inner section, × 400 diameter. Before use.

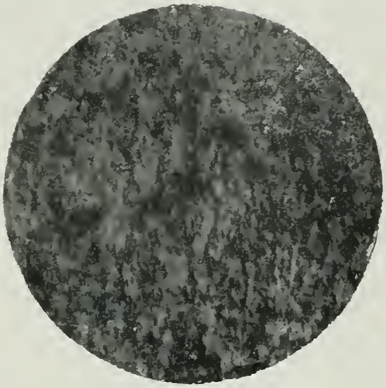


FIG. 26.—Outer section, × 400 diameter. After breaking.

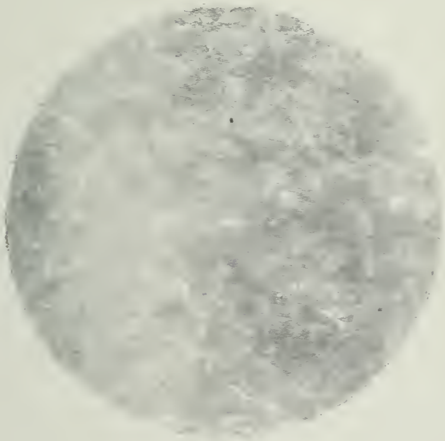


FIG. 27.—Middle section, × 400 diameter. After breaking.

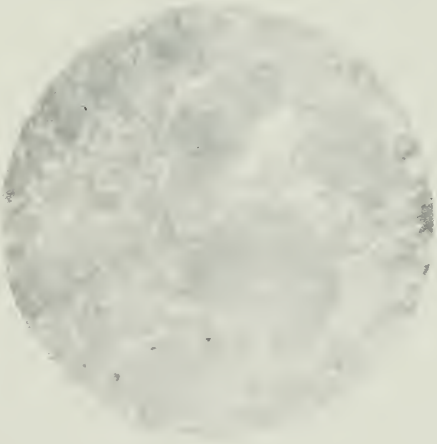


FIG. 28.—Inner section, × 400 diameter. After breaking.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. IX. JULY, 1908. No. 1.

Proceedings
AT
Ordinary General Meeting, July 18, 1908.

The Ordinary Monthly General Meeting of the Society was held in the Chamber of Mines, on Saturday, July 18th, Mr. R. G. Bevington, (President), in the chair. There were also present :—

50 Members: Prof. J. Yates, Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, C. B. Kingston, A. Richardson. C. B. Saner. G. O. Smart, Prof. G. H. Stanley, Jas. E. Thomas, A. Whitby, H A. White, Prof. J. A. Wilkinson, W. A. Caldecott, W. Cullen, E. H. Johnson, J. R. Williams, A. J. R. Atkin, W. Beaver, W. K. Betty, B. P. Carter, F. W. Cindel, A. A. Coaton, G. A. Darling, P. L. Edwards, A. D. Gilmour, J. Gray, J. H. Harris, B. J. Hastings, A. Heymann, H. R. Jolly, J. A. Jones, J. Kennedy, Hy. Lea, J. Lea, W. P. O. Macqueen, G. Melvill, H. Meyer, S. Morison, F. T. Mumford, T. T. Nichol, J. F. Pyles, O. D. Ross, C. E. Rusden, R. Stokes, J. A. Taylor, A. Thomas, A. D. Viney, J. Watson, A. Wilkinson and L. J. Wilmoth.

10 Associates: Messrs. J. Atkinson, J. Cronin, A. L. Edwards, O. Harrison, C. B. Hilliard, C. A. Robinson, B. Southwell, W. A. C. Tayler, W. E. Thorpe, and W. O. Williams.

15 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting as published in the *Journal*, were confirmed.

Prof. J. Yates introduced Mr. R. G. Bevington, the newly elected President to the Society, who thereupon took the chair.

NEW MEMBERS.

Messrs. Jas. Lea and W. Beaver were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows :—

Ackermann, Audley Harold, Resident Mining Engineer's Department, Bulawayo. Mining Engineer.

Barratt, Rowland Lorraine, Bushtick Mine, per Private Bag, Bulawayo. Assayer.

Bawden, Frederick Alexander, Knights Deep, Ltd., P. O. Box 143, Germiston. Mine Sampler. *(Transfer from Associate Roll.)*

Coats, Joseph, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Chief Sampler.

Colvile, Cecil Augustus Bradshaw, P. O. Box 669, Bulawayo. Mining Engineer.

Coulter, William M., Geldenhuis Deep, Ltd., Cleveland. Acting Cyanide Foreman. *(Transfer from Associate Roll.)*

Cumings, Charles Ernest George, P. O. Box 305, Bulawayo. Engineer.

Dowdle, Phillip John, East Rand Proprietary Mines, Ltd., P. O. Box 80, East Rand. Foreman, Cyanide Works.

Dyer, Sidney Charles, c/o the Resident Mining Engineer, Bulawayo. Mining Engineer.

Mageean, John Colquhoun, Knights Deep, Ltd., P. O. Box 143, Germiston. Civil Engineer and Surveyor.

Perkin, Dan Ward, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Assistant Surveyor.

Popham, John Leyborne, P. O. Box 669, Bulawayo. Mining Engineer.

Rogers, Nelson, Geldenhuis Deep, Ltd., Cleveland. Surveyor.

Stiell, William, Jumbo Mine, via Salisbury, Rhodesia. Mill Manager.

Stokoe, John Calvert, Pigg's Peak Development Co., Swaziland. Assayer and Surveyor.

Taylor, George M., Portland Mill, Colorado Springs, Colorado, U.S.A. Mill Superintendent.

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting :

Kirkland, Henry George, Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland. Clerk.

Parfitt, Walter, Nigel G. M. Co., Ltd., P. O. Box 20, Nigel. Cyanider.

Roy, James, Nigel G. M. Co., Ltd., P. O. Box 20, Nigel. Cyanider.

Tipper, Loton, Nigel G. M. Co., Ltd., P. O. Box 20. Nigel. Cyanider.

It was notified that letters expressing regret for inability to be present had been received from Mr. R. N. Kotzé (Government Mining Engineer). Prof. J. Orr (President, Transvaal Institute of Mechanical Engineers), Mr. A. C. Holtby (Vice-President, S. A. Association of Engineers), and others.

CORRESPONDING MEMBERS OF COUNCIL.

The President stated that the Council had elected the following Corresponding Members of Council :—

Mr. R. B. POLLITT (Durango) for Mexico.

Mr. G. M. TAYLOR (Colorado Springs) for the United States.

Mr. ARTHUR YATES (Sumatra) for Asia.

Further appointments for other parts of the world would be made at an early date.

Mr. J. Gray (*Member*): In looking over the last *Journal* I noticed an omission which was probably due to the interest taken in the discussion between Mr. Laschinger and Prof. Wilkinson on what General Botha calls "co-operatie." We forgot to tender our usual thanks to Mr. Rowland for his services as Secretary. We all appreciate his work, and I feel that it is due to him to a large extent that the Society holds the position which it does to-day. I therefore move a vote of thanks to Mr. Rowland.

The President: I should like to inform you that the Council has passed a vote of thanks to Mr. Rowland, and also voted him a bonus for his services. The matter you mention has not been overlooked by me, and you will see that, personally, I make some reference to the subject in some remarks which I shall make later on.

INAUGURAL ADDRESS.

By THE PRESIDENT
(R. G. Bevington, M.I.M.M., M.Am.I.M.E.).

I must first thank you for the honour which you have done me, and the confidence you have reposed in me in accepting me as your President for this coming year, and you may be sure that all I can do to further the interests of the Society will be done. I have been a member of the Society from its earliest days, and have its well-being and well-doing at heart. Prof. Yates has just joined, what he has termed the "Old Guard" of Past-Presidents, and I am sure you will all unhesitatingly agree with me that the Society is greatly indebted to him for the excellent manner in which he has carried out his duties during his term of office. The zeal and energy which has been displayed by your late Council in the carrying on of the business part of the work of the Society, and producing our valuable *Journal* merits the highest praise, and I am very glad to see that it has pleased you to re-elect so many of its members to assist me during the coming year. Of the work of our Secretary, Mr. Fred. Rowland, one can only speak in terms of the highest commendation, and the Society is fortunate in possessing the services of one who is so keen in its interests.

It has been left to me by my predecessor to touch upon the work of the past year, the business aspect of which has been placed before you in your Council's Report.

With regard to the practical and scientific aspect, the members must have gained much knowledge from the number of instructive and interesting papers upon mining, milling, metallurgy, mine ventilation, stope drills, and so many other subjects which fall within the scope of the Society's work, which have been read before them. I am very pleased to see that we have had so many papers contributed by members in distant parts of the world, for that shows the hold which our Society has upon its members though far away, and the interest which they take in contributing papers upon subjects which they know will afford us food for reflection and healthy criticism, and I trust that the ensuing year will be as productive in this respect and even more so.

The increase in Mining members and the number of able papers on mining matters which have been given us has fully justified the course taken in adding the word "Mining" to the title of the Society. I feel sure that there are still many more mining men who will become members when they realise that the Society is not a closed one, but is open to all to whom the subjects dealt with are of sufficient interest to cause them to wish to join. It is through the interchange of views at our meetings, and in our *Journal*, that one's interest becomes carried into other branches of our work than that particular one with which one is perhaps specially occupied, and thus an incentive is given to learn more of the work being carried on by others and the difficulties which they have to contend with, which all tends to cultivate a broader and more tolerant frame of mind.

In reflecting upon the progress of mining and metallurgical work upon the Witwatersrand, having been associated with these fields from their very early days, it is most interesting to me to look upon the great advance which has been made in every branch of the industry. Magnificent headgears and huge hauling engines have taken the place of the old horse whims and whips which I remember. Large shafts carefully planned and proportioned to the work required of them have taken the place of open cuttings. Batteries of hundreds of stamps of heavy weight have supplanted the 10 to 20 light stamp mills, and in doing all this, the best talent and scientific knowledge obtainable has been brought to bear, with a result of which all who have been engaged in the work must feel justly proud.

The advent of the cyanide process marked a point from which the greatest strides in the metallurgical part of our work have been made, and this process has afforded food for deep thought and research on the part of our chemists with the result that the whole method of application has been most carefully studied and worked out, but that is not saying that there is no opening for further improvements.

Chemical researches in the matter of strength of solutions employed, the combating of cyanicides, and the treatment of zinc to obtain better precipitation, have resulted in replacing the strong solutions which were at first deemed to be necessary, by what then seemed to be solutions of such little strength as to be useless. This, assisted of course, by reductions in manufacturers price, has resulted in greatly reducing the cost per ton treated, of what is still the largest item in the cost of treatment, viz., cyanide. Researches have proved that these weaker solutions are as effective and indeed more advantageous in that baser metals, which may be present in the ore, are less readily taken up by a weak than by a strong solution, and in this connection it may be remarked that the more rapid, and at the same time thorough, the treatment can be made, the less the amount of base metal which will pass into the solution.

Slimes, the making of which we at one time strove to avoid, are now almost welcomed, and the evolution of their treatment in a satisfactory manner has added enormously to the quantity of ore of low grade which may now be treated at a profit, so much so that rock is now being milled, and yielding a very handsome profit, having a less original content of gold than the assay value of tailings from the same class of ore which were refused by a cyaniding syndicate many years ago, as not being of sufficiently high value to be worth treating. One must not overlook the process for re-treating old tailing dumps, nor the fact that solutions very weak in cyanide, but holding minute quantities of gold and which at one time were looked upon as waste, are now passed through extractor boxes before final disposal, in order to endeavour to recover every grain which may be recoverable.

Turning to present practice. Milling work on the Rand seems to have settled down to a great uniformity of practice in the conducting of the general operations, though, of course, the question of greatly increasing the weight of stamps is occupying the attention of some of our leading metallurgists, and in this connection I would refer to the new mill of the Simmer Deep G. M. Co., the weight of the stamps in which will be 1,700 lb. In this case will also be put to the test the advisability of dividing up a large mill into separate units of 10 stamps, each unit being electrically driven. I cannot but think that this scheme will prove an undoubted success. As you will doubtless learn more about this, one of the newest plants on these fields, from another source, I will leave the matter to be dealt with by those more able to do so than myself.

Cyaniding, too, would appear to be conducted upon generally recognised lines in the carrying out of the work, but still the energy of our members is always being devoted to the bringing out of some further improvements.

Slimes, which I have before referred to, as having been at one time looked upon as a bane and a bugbear, are now so successfully, and one may say so easily treated, that if the general grade of the ore on these fields were higher, it is quite a question whether the practice of all sliming would not be resorted to. In the near future I think we shall see some improvements made in the means for handling sands and slimes, so as to make plants less cumbersome and extensive, thereby reducing the initial cost of same, and it seems possible with the many improvements which are being made in slime filters that the decantation process may be superseded, or at least greatly assisted, by filters of the pressure or vacuum type in order that the slimes residues may pass out of the works deprived to the fullest extent of their content of soluble gold.

Tube milling, which on its introduction met with many objections, seems to have held its own and with very beneficial results upon the gold production and the lowering of the value of the final residues, but I would suggest that there is yet work to be done in discovering and defining the most suitable size and shape of the mill and the correct speed at which it shall be driven, also with regard to the diameter of the discharge outlet as governing to a certain extent the length of time which the pulp shall remain in the mill before making its exit therefrom. I am also inclined to think that a means should be provided for varying the number of revolutions as the internal diameter becomes greater through the wearing of the lining, for it does not appear to be theoretically correct that what is the correct speed for one internal diameter is correct for a different diameter. The use of tube mills as secondary crushers or rather grinders, has resulted in greatly increasing the duty per stamp owing to the possibility of using screening in the battery of larger mesh than would otherwise have been economically possible, and withal has reduced the value of the final residues.

In mining work there is always something new being done in the application of methods and overcoming the difficulties of getting the broken ore from the working places quickly to the surface.

We are all looking to the near future to give us a handy stoping drill, and in this connection I would refer to the forthcoming stope drill competition. This competition will, I am sure, be most closely followed by mining men not only here but in all parts of the world. The benefit of the results to this community, provided that a reliable, light, efficient, and economical stoping drill, and one which will not be costly in repairs, can be found, will be immense, as it is the cost of mining which tells so much against many low grade propositions. The conditions and regulations for the trials appear to have been drawn up with the utmost care, and it is hardly conceivable that any question or contingency can arise under them which has been overlooked and not provided for. The composition of the committee, both as regards the members nominated by the Transvaal Government and those nominated by the Transvaal Chamber of Mines, is such as to inspire complete confidence in those who may be competitors, and also in the whole of the mining community and the public, who will await the findings and awards with the keenest interest.

The metallurgy of gold is simple compared with the problems which beset the metallurgist who engages in the work of dressing base metal ores and winning their contents. Our country is without doubt very highly mineralised, and although not much has as yet been done in this direction there will rise up in the future a large base metal industry, and one would, of course, wish to see the youth of South Africa in a position to take an active and leading part in such an industry. This it will not be able to do without having gained its knowledge in a practical and thorough manner, and this is the advantage which the Transvaal University College will offer in its new building, by means of ore dressing and metallurgical laboratories, and I counsel all those who desire to be able to take their due part in the development of the mineral resources and wealth of the country to lose no time in undergoing a thorough course of tuition in this excellent Institution, under the direction of the able and energetic staff of professors which is associated with it. There is no place in up-to-date practice for the out-of-date worker. It is becoming more and more essential that a man should know both the theory and practice of his profession if he would compete with and rise above his fellow men, and I venture to say that men who have had a good technical and practical training will be more and more in demand to the exclusion of the rule of thumb worker.

With regard to the ore dressing and metallurgical laboratories, there will, of course, be certain items of plant, models, etc., required beyond those which are now possessed, and it is to be hoped that money for this purpose will be unstintingly provided, in order to place the College in a position to compete with oversea mining schools. It is also to be hoped that the ore dressing plant there may be thrown open to all who may be competent to use it and by this means assist prospectors, miners, the mining houses, etc., in short, be of benefit to the country generally. At present ore is being sent out of the country to be treated, but several small plants are now, or are about to be, erected in order to treat the ore on the spot.

That the base metal possibilities are receiving attention, and that the output from the various sources is growing, may be seen from the report of the Government Mining Engineer for the year ending June, 1907, wherein it will be found that in the year 1906-07 base metals were produced of an estimated value of over £90,000 as against £19,500 in 1905-06, and I trust that in the next report a further increase will be shown.

The fine new building of the Transvaal University College, over which your retiring President, Prof. Yates, was kind enough to conduct me recently, is nearing completion, and I understand it is the intention to provide rooms for the offices of the various technical societies. This, I consider, will be a very good thing, as it will bring us all together under one roof, and will, doubtless, lead to the establishment of a common room, open to the members of the various technical societies, where technical papers and the journals of similar societies all over the world will be on view, and such meetings of scientific men on a common plane will assuredly lead to that co-operation among scientists of every branch which is so desirable and necessary for the progress of science in general, and which should eventually make for that Closer Union of all the Scientific Societies in South Africa, which is so much to be desired.

Apprenticeship to Mining.—For those who would take up the work of actual underground mining as a means of livelihood, the system of apprenticeship, as started by the Association of Mine Managers, would seem to be an excellent scheme, and it was with great regret that I learnt from some remarks made by Mr. Lane Carter at one of our recent meetings that "so far it had not been a success, and that at the present time there are very few apprentices underground." This is not as it should be. Means must be found to induce young men to submit themselves to a regular and thorough apprenticeship below ground, as is done by mechanics above ground, so that when a youth having duly served his term, applies for work as a full-fledged miner, the employer will have a guarantee that the applicant does possess a knowledge of the work he wishes to undertake, and to such as take up a serious view of mining work and wish to learn thoroughly what is most necessary, namely, the theory as well as the practice of the work, I would commend the evening classes in mining, which are held at the Transvaal University College, where the miner is taught how to work intelligently, and thus benefit himself, the mine, and also his fellow workmen. Under the new scheme of the Mine Managers' Association an inducement is held out to apprentices to attend these classes, which I find, on enquiry, have a large membership, and which are attended by men coming from far distant parts of the Rand, showing at the same time the attraction of the classes and the keenness of the students. The youth of this country has to learn to appreciate the fact that if it wishes to take a proper part in the development of the resources of this land it must take measures to submit to a course of study, not alone in one particular subject but in a general knowledge of science, and to the youth I would say, the means are at hand, make use of them.

I look to the Metallurgical Trials' Committee, which has lately been formed under the ægis of the various Mining Houses, for much assistance to the mining and metallurgical world. Co-operation of this nature will permit of far fuller and more practical investigation of the merits of any invention or improvement than would otherwise be the case, and I am sure that all those who have thought out improvements, which they are individually unable to have tested or experimented upon, will note the constitution of this Committee with pleasure, and will not be afraid to lay their ideas before the representatives, as I am confident that all trials made will be carried out in the most thorough manner.

Agriculture and Agricultural Chemistry.—There is little question that, apart from the mining industry, which is, undoubtedly, at the present time the staple industry of the country, there are enormous possibilities in the direction of agriculture, and owing to the many varied natures and constituents of the soils to be dealt with, the problems of properly dressing and manuring the land will provide much work for the chemist, without whose aid I fear those who engage in the farming industry will be but floundering in the dark. Those who read that most interesting production, the *Transvaal Agricultural Journal* (which I may state is free to every citizen of the Transvaal for the asking), will appreciate the great amount of work which has been done in every branch of agriculture by those employed on the Government farms, and I trust that our Government will always realise the great value of the various researches which have been made in the past, and the value of which will perhaps be more apparent in the future, to those engaged in agricultural pursuits. We know that there is a prevalence of a great number of various descriptions of pests, the occurrence of which dishearten the farmer and often indeed stultify his efforts, but I am sure that by perseverance and careful study the difficulties and dangers from this cause will eventually disappear, and in the attainment of this end the chemist will figure largely. I again venture to hope that our Government will recognise the great work which has already been done, and which still has to be done, and will give every encouragement to foster scientific investigations, and the work of the chemist in its application to the benefit of what must become one of the great industries of this vast country with its many and varied conditions of climate and soils.

South Africa should, undoubtedly, be in a position to become a country absolutely self-supporting in every respect. The question of irrigation and the proper conservation of water is one which may rightly and safely be left in the hands of our brother scientists, the engineers.

We have as a lesson in possibilities our marvellous gold industry, and in this respect I cannot do better than refer to the words of that eminent financier, Herr Dernburg, who recently visited us. In the course of a most instructive and eloquent speech he stated that there was a time when the standard of currency was greatly exercising the minds of financial men in the great centres of the world, the question being whether the standard should be gold or silver. At that time arose the Goldfields of the Transvaal, the output from which settled the question once and for all; it was to be gold. I doubt if this important fact had ever before been made patent to the world (outside the world of financial men), and I take it to be a happy augury that if South Africa has been able to settle this important question she will be able to settle many others of as great importance to the welfare of humankind.

Our Society continues to be on terms of the greatest amity with the kindred societies; this is much evidenced by the fact that some of our members have been invited by the Transvaal Institute of Mechanical Engineers to assist upon a committee, which that Institution has formed for the purpose of establishing South African standards for strengths of materials, etc.

Our Society is now widely represented by members in all parts of the world, but there is still much further scope for its usefulness, and I trust that those who have been and will be appointed (under the new scheme, which you have approved of and which strikes me as such an excellent one) "as Corresponding Members of

Council" will do their utmost to further our interests by bringing in new members, and by obtaining for our information and instruction, papers upon the many and varied subjects which come under the scope of our work, and also help us by contributing to our discussions. While on the matter of discussions, I cannot too strongly urge upon members the value of the fullest possible discussion upon a paper, as by this means we learn the views of individuals, and obtain the results of the experience of many hard and practical workers who may not feel able to write original papers, but whose attention and interest is awakened by statements made in papers written by others, which statements they may be able to confirm or otherwise from their own experiences, and thus the discussion becomes as valuable as the original paper. I would also urge our members to come forward with descriptions and records of processes and methods which they may have tried and not found to be successes. The information contained in a record of trials which have proved failures is of the greatest value to the metallurgical world.

The reduction in working costs, improvements in plants, and higher efficiency in the mining and metallurgical departments, as well as better extraction, have brought large bodies of ore, which were in former times classed as "unpayable," either within, or very nearly within the "payable" class, and it must be the aim of all of the members of our Society, both the mining and the metallurgical members, to take advantage of all existing improvements and to endeavour to devise new ones in order to assist in bringing down the general costs of production to such a point that aided, we may hope, by still better extraction of the gold, it shall be possible to see a profit where but loss existed before. In this endeavour we must work hand in hand with our good friends the kindred societies of engineers, whose aims run on lines parallel to our own. Thus can we do our portion towards making it commercially possible to open up new mines and to reopen many old ones, to the end that the industry may be able to sustain a larger white population, and that by such means the burden of those who are now suffering so severely from want of employment may be lightened, and prosperity may again reign in this country.

Mr. A. McA. Johnston (*Vice-President*): It is my very pleasant duty to-night to propose a very hearty vote of thanks to our President for the very able and interesting address which we have just listened to. His résumé of the work of the past year has been a masterly one, and the careful hints which he has thrown out are sure to result in some interesting discussion in the future.

I think the Society is to be congratulated on having Mr. Bevington as President. His work the Council know very well. As long as I can remember I have always looked upon Mr. Bevington as one of the men who was bound to come forward, and now that he is President we are very much to be congratulated. His attention in the past has been directed to the very important matter of the financial state of the Society, and the present favourable position of the Society in this respect is very greatly due to the efforts which he has put forth. In proposing this vote I can only assure Mr. Bevington that he has the whole Council with him. In the coming year the Council will do all that they can to help Mr. Bevington to make this Society the success it has been in the past. And the members will greatly help the Council if they also assist in this direction.

Mr. W. Cullen (*Past-President*): I should like to associate myself with all the remarks which have fallen from my friend, Mr. McArthur Johnston. I think that we, as a Society, are to be congratulated on having Mr. Bevington as President for the coming year, and I am quite sure, from what I have heard from one source and another, that he will have a most successful year of office. If anyone has won his spurs, Mr. Bevington has. I think I may safely say that he has done more solid work than many of us put together. I heartily second the vote of thanks.

The President: I must thank you very much for the vote of thanks, and for the kind words which have fallen from the proposer and seconder. I am afraid you are crediting me with rather more than is my due, but I shall always try and act as if I had done the good things which you have credited me with.

BATTERY AND CYANIDE GOLD SMELTING.

By A. Thomas (Member).

As it is some time since the subject of gold smelting has been discussed by this Society, the following notes are written with the object of giving a brief outline of some present practices, and of providing an opportunity during discussion, for members to bring forward other methods at present in vogue.

The separation of the mercury from the gold is done in cylindrical retorts, made of cast steel, measuring 53 in. long, 12 in. in diameter, and 1½ in. thick. The retorts rest on two water-cooled bearers, with the usual water jacketed condenser for the mercury fumes. The flue of a retort furnace has three openings, measuring 11 in. × 8 in., arranged above the retort one over each end, and the other over the centre, thereby insuring a uniform temperature over the whole body of the retorts. This arrangement of the flue is necessary, as with the single flue retort furnace one part of the retort is generally heated too strongly, whilst other parts are insufficiently heated to volatilise the mercury completely. The capacity of this retort is 5,000 oz. of amalgam, and in charging, semi-circular trays are used. These are whitewashed inside before receiving the amalgam, and care is taken not to fill the trays too full. The doors of the retort are sealed with a joint made of asbestos cord soaked in water. As soon as the mercury begins to come through the condenser the fire is checked and a uniform temperature maintained for about two hours, or until the mercury ceases to come off. The temperature is then raised to expel the remainder of the mercury.

If the temperature is too high in the early stages of retorting, the distillation will be too violent, and spluttering of amalgam in the retort is likely to ensue. The time required for retorting is from 4 to 4½ hours.

After removal from the trays, the gold is weighed into 1,000 oz. lots and smelted in No. 40 plumbago crucibles; a little borax is added as the gold melts down, but this is removed by skimming, after being thickened by the addition of a little fine sand. The bullion is afterwards cast into bars, which show a loss in weight of about ·72% on the retorted gold. The mercury is afterwards weighed and returned to the mill. The difference between the weight of the amalgam charged into the retort and the weight of gold and mercury recovered is about ·14% on weight of amalgam. Several assays of the retorted mercury have been made, and in each case only a trace of gold found. The coal consumption for retorting 5,000 oz. of amalgam is about 350 lb.

The fire boxes used for calcining the zinc gold slimes are rectangular in shape, measuring 44 in. × 22 in. A tray made of 2 in. cast iron with three sides turned up 1½ in. is placed over the fire box. The trays in which the slimes are calcined are placed over these furnaces and are made of ⅛ in. mild steel, 48 in. long, 26 in. wide and 8 in. deep. Before receiving the slimes these trays are whitewashed, as this prolongs their life and prevents the slimes from adhering to the bottoms, especially if the temperature has been too high during the calcination process. Great care must be taken in carrying out the calcination, as upon this the success of the smelting operation depends. The temperature is kept low at first and gradually raised as the operation proceeds. As a rule, about three hours is required for a successful calcination. The usual loss on this operation is about 42·4% of the original weight. Tests seem, however, to show that the bulk of the loss is due to moisture, which may run up to as much as 38%, the remaining 4% may be due to the removal of carbonaceous matter, and other changes, which may cause a loss of weight. A curious feature of the calcined slimes is the presence of a purple colour, which would seem, though I am not positive on the point, to vary in intensity in proportion to the richness of the slimes. According to Muller (see *Metallurgy of Gold*, by T. K. Rose, p. 35), gold in a very finely divided state gives a purple colour in the presence of alkalies and alkaline earths, and it may be that this purple colour is the same as mentioned by Muller. On completion of the calcination the tray is removed from the furnace, and when cool, taken to an enclosed room, where the calcined slimes are weighed and fluxed. Every precaution should be taken at this stage to avoid unnecessary handling and consequent dusting of the slimes.

The fluxing is carried out as follows:—The calcined slimes, which are more or less caked, are weighed direct from the trays without any grinding whatever. Plumbago crucibles No. 100 with clay liners are used for smelting. The slimes are weighed and placed in the pots with the required amount of manganese dioxide. The pots are only filled to within 4½ to 5 in. from the top, the rest of the space being used for the other fluxes, composed of borax and silica, old crushed liners being used as a source of silica. The following may be taken as a representative charge: Calcined slimes, 100 parts; manganese dioxide, 7·5 to 10 parts; crushed liners, 25 parts; fused borax, 20 to 25 parts.

Naturally, the flux is used within certain limits, according to the quality and physical features of the slimes after calcination. All new plumbago crucibles are subjected to a preliminary treatment before use, thus increasing their life to a considerable extent. The treatment is as follows:—They are placed in the furnace and heated for 30 minutes, to remove the glaze, after cooling they are placed for 45 minutes in a saturated solution of borax, and are afterwards dusted over with powdered borax and placed over the calcining furnace to dry, and when dry, reheated in the reverberatory furnace until the borax fuses. New clay liners, whenever possible, are used in the first charge of the smelt. This obviates the necessity of suddenly subjecting them to the high temperature of the furnace prevailing with subsequent charges.

A type of furnace which, with a few slight alterations, would be quite suitable for this work, is the old reverberatory pan furnace, known locally as the Tavener furnace ; one point I may mention in connection with furnaces used for smelting rich gold bearing materials, is the lack of proper condensing chambers in the flues. I am convinced that some kind of condensing chamber is necessary and should be attached to all furnaces smelting finely divided and rich product, such as cyanide gold slimes. Even with all the precautions that can be taken in fluxing, such as using a borax cover, or placing covers on the pots, I have every reason to believe that gold is carried off to the flue, both mechanically and by volatilisation.

The buttons of gold obtained from the fusion are usually of high grade, and ready for casting into bars. Occasionally, a thin skin of matte may appear on a few of the buttons, and when this occurs they are arranged so that those with the matte are placed in the same crucible as the highest grade button of the smelt, thus averaging up the fineness in the bars. No attempt is made to desulphurise the matte in smelting, but this is skimmed off and kept for future treatment with the bye-products in the pan furnace, where it will be desulphurised with battery screens. This matte generally occurs in case of cracked liners when the slag comes in contact with the graphite crucibles. My experience goes to show that to a very large extent the fineness of the bullion produced on these fields depends mainly on good calcination. Perfect calcination has two other great advantages, firstly by reducing the bulk of the material, secondly, by perfect oxidation of the base metals, doing away with the necessity of using a large quantity of oxidiser in fluxing, thus reducing the value driven into the slag—particularly of the silver.

For very base bullion, owing to the presence of metallic lead and copper, I find that cupellation with clean lead is the only satisfactory method of bringing up the grade. This, however, should not be necessary on these fields, as low grade bullion is rare and unnecessary.

In casting cyanide bars it is usual to keep their weight in the neighbourhood of 700 oz. A point that has often been raised on these fields, is to what degree of fineness should bullion be brought. At present we have all kinds of pains and penalties to avoid at the hands of the refiners in London. This seems rather extraordinary when we consider the tonnage, for that is what it amounts to, of the gold we turn out. A mine that refines its gold up to over 900 parts per 1,000 fine receives no better terms than one which just reaches the mark of 900 gold and silver. There is no doubt that every penny we spend on refining our bullion beyond the point necessary to escape the penalties is, to my mind, unnecessary. The present system is certainly not satisfactory, and would bear thorough investigation at the hands of those who have the disposing of the gold on behalf of the mines.

Another point also suggests itself, why should we ship all our bullion home for refining? I can see no reason why a refinery should not be established on these fields. The technical part of the work can be successfully carried out here, but I will leave the discussion of the financial side of this question to members who may have more data at their disposal.

For the treatment of bye-products many mines are equipped with the usual kind of reverberatory pan furnace. As the construction of the furnace is the main factor which governs efficiency and working costs. I give the following dimensions of a furnace of this kind, hoping they may be of use to some of the members.

The pan in which the furnace hearth is built is made of ½ in. iron, measuring 9 ft. × 6 ft. with a depth of 24 in. :—

Size of grate in fire box, 3 ft. 6 in. × 3 ft. ; from fire box of grate to top of roof, 2 ft. 4 in.; top of bridge wall to roof, 11½ in. ; size of furnace hearth, 7 ft. 6 in. × 3 ft. 6 in. ; bridge end of hearth to roof, 2 ft. 5¾ in. ; centre of hearth to roof, 2 ft. 6 in. ; lowest point of roof to hearth, 1 ft. 7 in. ; the hearth of the furnace has a fall from all points to tap hole of 3½ in. to 1½ in.; uptake flue from furnace, 2 ft. 6 in. × 9 in. ; capacity of furnace for lead bullion 4,200 lb. = 61,268 oz. troy.

The bye-products consist of the slag from the smelting of gold zinc slimes in the reverberatory pan furnace, ground up crucibles, liners, etc. In treating these bye-products, the aim is to produce a liquid slag of low specific gravity, and thus reduce as much lead as possible from the materials treated. This is done by the addition of metallic iron in the form of battery screens. As a precaution against any residual high values of the precious metal in slag, the charge, prior to withdrawing the slag, is given a wash down with litharge and coal dust.

All variations in the charges smelted are regulated by experience. It would be very interesting to know by analysis the exact composition of the slag and other materials to be treated and so be able to flux from a theoretical standpoint. However, as these parcels of slag and other bye-products are not very large, and the composition of them varies from time to time, an analysis would not furnish any definite information for the treatment of future bye-products.

The present method is crude, but the value of the residual slags shows that the results are satisfactory. Naturally, this method is only applicable in a case similar to the one we are dealing with, because fully 90% of the material treated has already been fluxed, and merely requires re-smelting and reduction of the metals. The principal material to be smelted being the slag from the zinc slimes smelt, this is always taken as a base from which to work. When there is a large quantity of refractory materials to be smelted, it is worked off in small lots. Should these present any difficulty, the amount of assay slag usually added to the charge is slightly increased, ground up liners, or any other gold bearing silicious material is added in small lots to each charge, as the slag from the cyanide smelt is nearly always basic, and some protection is needed for the furnace lining.

In commencing a smelt, a few bars of lead are placed in the furnace, so as to form a bath, and prevent the slag from corroding the bottom of the furnace. The first charge is made up and its behaviour in the furnace carefully noted, so that any defects may be remedied in the succeeding charges. It is sometimes found necessary to return the slag to the furnace for re-treatment on account of the high values contained therein. The following is a brief account of a bye-product smelt.

Amongst the materials treated were, reverberatory pan furnace slag, bullion furnace ashes, extractor house sweepings, old cupels, brick from furnace bottoms, calcining trays, battery screens, elephant cord from battery, assay slag, etc.

The mixture was made up into 14 charges of about 900 lb. each, and contained 9% litharge and 5·5% iron, in the form of battery screen and old calcining trays.

The average value of the slag drawn off, assayed as follows :—

Gold, 3·99 dwt. per ton ; silver, 337·59 dwt. per ton ; lead, 6·5%.

The weight of lead bullion recovered = 19,143 oz. troy. The total litharge added only accounts for 12,711 oz., thus showing a gain of 6,432 oz. After deducting 797 oz., the weight of the gold and silver recovered by cupellation, there remains a surplus of 5,635 oz. of lead, which has been recovered from the material treated.

The lead bullion after cupellation yielded the following bar :—

Bar No.	Bullion oz.	Gold Fineness.	Silver Fineness.	Oz. Fine Silver	Oz. Fine Gold.
C 740	840·41	477·5	471·5	396·250	401·290

In this smelt the last few pots of slag drawn off are put aside for treatment in the next smelt, since fine shots of lead may be skimmed off. Before tapping the lead bullion off, the metallic lead is concentrated and purified by oxidation and the dross afterwards taken off. In this particular smelt the dross carried 28% metallic copper, the remainder being oxides of lead and finer shots of metallic lead. The cupel used in this smelt was made of bone ash, manufactured on the mine. The bones were collected from the kitchen at the native compound, and burnt in the cupel furnace, an iron plate being inserted on the cupel so as to form a bottom for the furnace. The burning presented no difficulty, and the calcination was complete in four hours. It is most important that the bones should be completely burned, so as to expel all animal and carbonaceous matter. The loss in weight during calcination is about 54%. The bone ash obtained was equally as good, if not better than the imported article, at less than half the cost. In preparing the bone ash for making the test, I used 14 lb. caustic potash in solution to 400 lb. bone ash with satisfactory results.

In presenting these brief notes I do so with the hope that the members engaged in this class of work will raise a discussion which will serve to bring forward a few of the newer innovations in smelting on these fields, and so bring our published information up to date.

The President : I think the thanks of the Society are due to Mr. Thomas for having so lucidly placed this matter before us, and I have no doubt that his paper will lead to a good deal of discussion, and through that discussion we shall, as he says, be able to bring the information on this subject up to date.

Mr. C. W. Lee (*Member*) : In seconding the vote of thanks to Mr. A. Thomas for his paper, I think that the author has brought up a few points which will produce some lively discussion. In bringing forward a paper like the one just read, I regret very much that those little difficulties which crop up, and have to be dealt with at a few moment's notice, have not been touched upon. At times the assayer has to cast a bar and send it to the bank at one or two hours' notice, and should the bullion not be of the required fineness, he has not always at his command a cupellation furnace to fall back upon to bring up the quality of his bar, and has, therefore, to resort to other little remedies, which I regret the author has not referred to, but which the assayer has to be acquainted with all the same. I congratulate Mr. Thomas on his paper, and hope that at some future time he will go more fully into the subject.

SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

(Read at January and March Meetings, 1908.)

By W. A Caldecott, B.A., F.C.S. (Past-President.)

REPLY TO DISCUSSION.

Mr. W. A. Caldecott: In replying to the discussion upon my paper I must express both my appreciation of the leniency of the criticism, and my regret that none of our members actually engaged in silver ore treatment have thrown further light upon the points raised. It appears as if our corresponding members of council in Mexico will have a wide field for missionary effort in inducing the metallurgists of that country to share in the frank discussions, which have become so honourable a tradition in our Society.

Prof. Wilkinson and Prof. Stanley have corroborated certain features noted in the paper by giving us brief accounts, taken from recent technical periodicals, of the plants and methods on individual mines in Mexico. To their request for further information I can only plead the very limited knowledge of a passing traveller, and trust that some of our Mexican members will authoritatively reply to these queries. I may, however, point out that, as stated in the paper, the methods therein described apply to ores carrying both arsenic and antimony as well as silver sulphide, but that the concentration referred to doubtless separates a large proportion of these base metal compounds. Since the chemistry of silver cyaniding is a problem so complex and so little investigated, I may venture to express the hope that our Proceedings will, ere long, be enriched by a discussion of the subject, from the standpoint of modern science and experiment, by those representatives of the Transvaal University College who, in spite of the many calls upon their time and energy, have already done so much for our Society.

Mr. S. H. Pearce's remarks are of peculiar interest from the historical aspect, as evidencing the close relationship between Rand and Mexican practice. Many features of the old Rand Central Ore Reduction Company's plants, such as mechanical stirring gear, are much in evidence in Mexico to-day, and have a common origin in the views of Mr. Chas. Butters, our Past-President, and the large staff, including Mr. J. Johnston and Mr. E. M. Hamilton, whom he took with him from these fields. Mr. Pearce's early suggestion to employ lead salts has certainly become common practice in Mexico since his visit in 1904, but without discussing the extremely nice point of the relative solubility of the two remarkably insoluble bodies, lead sulphide and mercurous sulphide, I can hardly accept the lead sulphate hypothesis. Lead sulphate readily dissolves in excess of caustic alkali, and conversely addition of a soluble sulphate to an alkaline plumbite solution does not produce a precipitate. The suggestion in the paper that the oxide of lead exists in solution as an alkaline plumbite, or as part of the acid radical, serves to explain both of the above experimental observations.

It is gratifying that Dr. Moir's mathematical examination, from the standpoint of the ionic theory, of the experimental figures quoted in the paper has not revealed any serious discrepancies, and his contribution confirms and extends the general conclusions expressed. His theoretical deduction that excess of alkali is detrimental to the dissolution of the silver, by causing the existence in solution of Na_2S instead of $NaSH$, is not borne out by the results quoted in the paper (see this *Journal*, March, 1908, p. 268), but the tests given later indicate a prejudicial effect of alkali which may be ascribed to its solvent action on mineral sulphides in the ore.

Since the paper was read I have had various tests carried out on a mixture of oxidised and pyritic Mexican gold-silver ores from the El Oro, Pachuca, and Guanajuato districts. The analysis of the mixed ore is as follows:—

	Per cent.
Silica (SiO_2)	77·89
Metallic iron (Fe)	0·65
Ferric oxide (Fe_2O_3)... ...	3·39
Alumina (Al_2O_3)	5·73
Lime (CaO)	5·52
Magnesia (MgO)	1·33
Sulphur (S)	1·61
Carbonic acid (CO_2).	1·97
Water (loss on ignition) ...	2·02
	100·11

The ore assayed 16·3 dwt. gold per 2,000 lb. ton (equal to 27·9 gm. per metric ton*) and 490 dwt. of silver per 2,000 lb. ton (equal to 840 gm. per metric ton). The metallic iron shown in the above analysis was separated as such by magnet, and was, no doubt, introduced during fine-crushing on the iron bucking board. Part of the iron reported as oxide was probably combined with sulphur as pyrites.

All experiments were made on ore dry-crushed through a screen of 200 holes per linear inch. Hence the crushed ore was practically slimes from a metallurgical standpoint. By long boiling first in strong nitric acid, and then, after washing, in *aqua regia*, an extraction of 93% of the gold and 91·4% of the silver was obtained. Such a treatment would, of course, dissolve any base metal compounds, as well as the gold and silver, which the preliminary fine-crushing had exposed to the action of these powerful solvents.

---

* One gram per metric ton is equal to seven-twelfths of a dwt. per 2,000 lb. ton.

To test the possibility, indicated in the paper, of reprecipitation, by soluble sulphides derived from the ore, of the values in solution, the following experiment was made: 100 gm. of the above finely crushed ore was mixed with 200 c.c. of sodium cyanide solution of 0·05% strength (expressed in terms of potassium cyanide) and 1 gm. of lime, and the mixture shaken at intervals for six hours. The pulp was then halved into (*a*) and (*b*). The half (*a*) had its solution, which contained free cyanide, at once filtered off and assayed. The half (*b*) was put aside in a small closed flask, which it almost completely filled, for six weeks and shaken once weekly; at the end of this time the solution, which then contained no free cyanide, was filtered off and assayed. The results were as follows:—

TREATMENT.	GOLD DISSOLVED.			SILVER DISSOLVED.		
	Per ton of ore.			Per ton of ore.		
	Dwt per 2,000 lb. ton.	Gm. per metric ton.	% extraction.	Dwt per 2,000 lb. ton.	Gm. per metric ton.	% extraction.
(*a*) 6 hours' treatment	10·00	17·1	61·3	155	266	31·6
(*b*) Ditto + 6 weeks cont.	0 50	0·9	3·1	21	36	4·3

This showed that very serious reprecipitation takes place under these extreme conditions involving destruction of all free cyanide present, but a similar test with 0·25% solution showed practically no reprecipitation after 66 hours' contact, when the residual free cyanide strength was 0·21%. The possibility, however, of reprecipitation under certain conditions may partly serve to explain why such short and imperfect settlement in decantation slime treatment is practised in Mexico, if experience there has shown that prolonged and undisturbed contact between slimes and solution results at times in higher residual values through reprecipitation. Tests of a week's contact of ore submerged in cyanide solution and only shaken once daily gave poor results, so that favourable conditions rather than prolonged treatment seem to be the essential requisite. In general, there would appear to exist the opposing influences of solution, aided by the aeration due to agitation; and of precipitation of dissolved gold and silver favoured by absence of oxygen and destruction of free cyanide. A similar condition has been noted in regard to gold precipitated on zinc shavings in contact with cyanide solution.* The practical conclusion appears to be that separation of solution from solids should be prompt after the gold and silver is dissolved.

The following tabular statement illustrates the results of one series of experiments among others carried out on the above ore crushed through 200 mesh, and mixed with 77% of its weight of precipitated silica, on account of limited quantity of material available. The results given are the means of duplicate tests, which consisted, except where otherwise stated, in agitating by hand at ordinary temperatures 65 gm. of the mixture with 130 c.c. of a solution of sodium cyanide 0·25% strength (expressed in terms of KCy) in 800 c.c. closed vessels for six hours, together with the percentage stated in the table on the weight of the ore of the reagents mentioned. The time of treatment was purposely limited so as to exhibit the difference in rate of dissolving, as otherwise given sufficient time the less favourable treatment might equal the more favourable in the final result. The mixed and diluted ore thus experimented on had a value of 9·2 dwt. gold per 2,000 lb. ton (equal to 15·8 gm. per metric ton), and 277 dwt. silver per 2,000 lb. ton (equal to 475 gm. per metric ton). In (*e*), 0 5% solution was used, of which nearly half had been converted into HCy by titration with HCl beforehand. In (*h*) and (*j*) the lead salt had been dissolved in a minimum of caustic soda solution before addition to the ore. After agitation in each case was completed, the solution was at once filtered off, and the calculations are based on its gold and silver contents, rather than on the residue which might have decreased in value during the aeration consequent on thorough washing. Considerable free cyanide was present in the solution after treatment in each case. The following are the results:—

GOLD.

Treatment and Reagents.	Dissolved per ton of ore.		% extr'c-tion.
	Dwt. per 2,000 lb. ton.	Gm. per metric ton.	
(*a*) NaCy sol. + 0·1% BaO_2 + 1% CaO	8·25	14·1	89·7
(*b*) NaCy sol. + Oxygen (at 100 lb. pressure) + 1% CaO ...	8·20	14·0	89·1
(*c*) NaCy sol. at 58° F. + 1% CaO	7·65	13·1	83·1
(*d*) NaCy sol. at 100° F. + 1% CaO	7·60	13·0	82·6
(*e*) NaCy sol. + HCy sol. ...	7·15	12·2	77·7
(*f*) NaCy sol. + 0·1% PbO_2 + 1% CaO	6·75	11·5	73·3
(*g*) NaCy sol. + 0·11% PbO_2H_2	6·75	11·5	73·3
(*h*) NaCy sol. + 0·1% PbO_2, $x Na_2O$ + 1% CaO ...	6·35	10·9	69·0
(*i*) NaCy sol. + 0·11% PbO_2H_2 + 1% CaO	5·35	9·2	58·1
(*j*) NaCy sol. + 0·1% PbO, $x Na_2O$ + 1% CaO	4·50	7·7	49·0

---

* See this Society's Proceedings, vol. ii., July, 1899, p. 746.

SILVER.

Treatment and Reagents.	Dissolved per ton of ore.		% extraction.
	Dwt. per 2,000 lb. ton.	Gm. per metric ton.	
(*b*) NaCy sol. + Oxygen (at 100 lb. pressure) + 1% CaO ...	220	377	79·4
(*d*) NaCy sol. at 100° F. + 1% CaO	215	369	77·6
(*a*) NaCy sol. + 0·1% BaO_2 + 1% CaO	215	369	77·6
(*i*) NaCy sol. + 0·11% PbO_2H_2 + 1% CaO	195	334	70·4
(*h*) NaCy sol. + 0·1% PbO_2, xNa_2O + 1% CaO ...	185	317	66·8
(*c*) NaCy sol. at 58° F. + 1% CaO	180	308	64·9
(*f*) NaCy sol. + 0·1% PbO_2 + 1% CaO	170	291	61·3
(*j*) NaCy sol. + 0·1% PbO, xNa_2O + 1% CaO	145	249	52 3
(*g*) NaCy sol. + 0·11% PbO_2H_2	140	240	50·5
(*e*) NaCy sol. + HCy sol. ...	110	188	39·7

In studying the foregoing tables, it is at once apparent that the percentage extraction of the gold is usually better than that of the silver, but that a satisfactory gold result by no means implies an equally good silver extraction. Since the descending order of percentage extractions for gold is different to that for silver in the same experiments it follows that the same factors affect the two metals differently.

In general, the chief influence favouring ready solution of the gold is oxygen in some form, and the results are better according as this element is amply provided, or worse as substances are formed which consume it. The two highest extractions (see *a* and *b*) are due to the oxidiser BaO_2 or compressed oxygen. Variation in temperature (see *c* and *d*) shows little difference, possibly due to the alkaline CaO in the warmer solution dissolving more oxygen-absorbing sulphides, and thus counteracting any tendency of the gold to dissolve faster. The presence of HCy (see *e*) gives somewhat poorer results, which may be attributed to the presence of some gold alloyed with silver in the ore, and the slow rate of dissolving of the silver retarding the solution of the gold. The trials with lead salts all show poorer results than the foregoing. PbO_2 with caustic soda is worse than without (see *h* and *f*), and similarly in the case of PbO (see *j* and *i*). This may be attributed to the caustic alkali dissolving reducing sulphides from the ore. The available oxygen in PbO_2 causes, however, better results than the use of PbO (compare *f* and *h* with *i* and *j*); PbO without lime (*g*) is better than with lime (*i*); here also the solvent effect of free alkali on mineral sulphides may be accountable. The amount of lead available was in every case small as compared with the sulphur in the ore, but the general poor effect of lead is difficult to explain, except possibly on the grounds that the PbS formed absorbed oxygen more readily than a corresponding amount of Na_2S in solution.

The dissolution of silver is not only more imperfect than that of the gold, but is also more readily affected by varying factors. Whilst accelerated by warmth and oxidisers, it is retarded, on the grounds detailed in the paper, both by presence of soluble sulphides, and even more of hydrocyanic acid. This last is strikingly evident in the two lowest results (see *g* and *e*) where HCy is present in the solution, either through addition of HCl beforehand, or through the effect of latent acid in the ore in the absence of protective alkali. Whilst aurocyanhydric acid ($HAuCy_2$) is relatively soluble, I am not aware that the corresponding silver compound exists at all. Hence the presence of some alkali appears advisable and, in any case, its injurious effect is not so apparent as with gold. The above results do not exhibit the benefit from the use of lead salts which might be anticipated from theory, so far as it is developed, and from their common use in Mexican practice. The reason is obscure, although the amount of lead salt used was much less than the sulphur equivalent in the ore, and it is possible that some unrecognised feature of the ore experimented on or of the tests employed differed materially from usual working conditions.

It is possible that the development of the use of warm cyanide solutions on these fields, recently discussed in reference to accelerated slime settlement by Mr. A. Salkinson before this Society*, may be advantageously paralleled in silver ore treatment in Mexico, although the extensive use of electric power there does not provide a cheap source of heat in the shape of exhaust steam. To carry the effect of heat to an extreme, some of the same fine-crushed ore was boiled with twice its weight of 0·26% cyanide solution and 0·1% lime for six hours in an open beaker, water being added periodically to maintain a constant volume. This resulted in 90% of the free cyanide being decomposed, and only 28·3% of the gold and 32·6% of the silver being dissolved.

The foregoing experiments, however, leave much to be elucidated in reference to the precise influence, especially upon silver extraction under working conditions, of the following factors:—(*a*) Time of treatment, (*b*) temperature, (*c*) strength and volume of solution, (*d*) oxidising agents and aeration by agitation, (*e*) lead salts,

---

* See this *Journal*, vol. vii., June, 1907, p. 403.

(*f*) presence of free alkali, or hydrocyanic acid, and (*g*) reprecipitation of values. Whilst in Mexico the mechanical features of fine-crushing, agitation, and filtering of slimes have been skilfully and profitably developed, the actual percentage recovery by cyaniding and its cost, still afford scope for improvement,* and there is little doubt that intelligent and systematic research, from a chemical standpoint, on the above lines, would be as well repaid as it would be full of interest.

In concluding this reply, I may confess that the tentative views and speculations therein advanced are necessarily based on experiments upon material which was unfortunately limited in amount, and without the great advantage of constant contact and long acquaintance with working conditions and observations. Moreover, in the vast mining area of Mexico, so diverse and complex are the ores, that generalisations are much restricted, and special conditions and problems have to be dealt with in each district and plant. Hence, whilst those with greater knowledge of the subject will doubtless justly dissent from some of the views expressed, yet I trust that a sufficient residuum of correct deductions may remain to justify this addition to the small amount of published literature upon the subject.

The President: Mr. Caldecott has been justly thanked for his original paper, and I think he should also be thanked for the splendid reply which we have just heard. He has also to be cordially thanked for the very large amount of original research work which he has carried out in this matter and which, I am sure, will be carefully read and studied by our friends in Mexico. I am sure it will give them food for reflection.

Mr. W. A. Caldecott: In thanking you, Mr. President, for your kind remarks, I should again like to state that Mr. McA. Johnston and Mr. F. W. Watson have borne the burden of actually carrying out the experimental work described in the paper and reply.

NOTES ON RAND MINING.

(Read at March Meeting, 1908.)

By Tom Johnson (Member).

DISCUSSION.

Mr. J. M. Phillips (*Member*): *Stoping.*—I do not agree with Mr. Johnson in his suggestion of "large stopes and more machines," unless it be on a flat reef when the "cars" can be run into and filled in the stopes. The average stope here is a very confined place to work in, and only a limited number of "boys" can be got to lash economically at one time, and I certainly do not think that the rock from ten to twelve machines could be shifted each shift without unduly interfering and obstructing the machine men. As the bulk of the rock broken lies parallel to and within 30 feet of the face and has all to be lashed in one direction, it is evident that you cannot put more than a limited number on at one time and, therefore, your output from the stope is limited by the amount of rock you can take away from it, and not by the number of machines you can get to work at one time. From four to six machines is, in my opinion, the most that can be economically employed in a stope.

I have no faith in the "rigging gang," as suggested. Every man prefers rigging his own machines to suit his own ideas of the position of his holes, and as no two men will rig up alike, so no man will get the same amount of work out of another man's rigging as he will out of his own. Another objection, is that seeing the rigging gang will only have a specified number of machines to rig up, they will endeavour to rig them up in the shortest time possible, regardless of position or security. The result will be that several of the machines will need taking down and re-erecting, thus causing loss of time and holes to the machine man. Let each man rig his own machines, but see that he keeps spare benches, and if through any cause he is unable to drill over his bench in time to blast at the end of the shift, let him leave the machine standing for his partner, who may have better luck and so be able to finish the bench left and then rig up and drill over another by the end of his shift. The policy of breaking the rock until the stope is full, is to be strongly recommended. This policy was carried out on the Henry Nourse mine in 1888, under the management of Mr. Jas. Whitburn, and therefore is no new idea, as far as these fields are concerned, but I fear the introduction of big mills has not allowed our managers to continue the system. The practice of underground sorting and stacking was in vogue on the Jumpers mine in 1888, under the management of the late Mr. Z. Williams, and I was surprised to see it brought forward as something "new," also the method of filling old stopes with tailings was employed in the early days. The late Mr. Chas. Gluyas, manager of the Jubilee mine, adopted this method, bringing the tailings from the old mill in the carts which conveyed the ore from the mine. Overhand stoping is certainly the safest, affording certain protection to the workers from loose rocks and falling hanging, but I prefer underhand stoping for machines, as then you have better footing for rigging up, and all the holes are water holes, which one cannot have either in overhand or breast stoping. The author says, "if there is anything that should be drilled into miners, it is the fact that where benches are long enough, the putting of burden on the holes by lengthening them is cheaper than the putting of the burden by placing the holes further from the face of the stope." Now this plan is good when there is a good hanging or footwall to break to, but I doubt its cheapness in a tight stope.

---

* For instance, the official report of the Guanajuato Consolidated Mining and Milling Company for 1907 gives the cost of cyaniding at 7s. 7½d. per ton, with an average residue worth 6s. 2d. in silver and gold, equal to an extraction of 68·97% from sands and 70·37% from slimes. (See *The Mining and Scientific Press*, May 30, 1908, p. 750.)

The average length of hole here is 6 ft., and the burden 2 ft. 6 in., and the average charge is four to four and a-half plugs of gelatine. Taking, for instance, a bench 10 ft. long by 6 ft. wide, which will come with four holes, I take it the author would drill his holes 7 ft. to 8 ft. long instead of the ordinary 6 feet, and would thus have a burden of 8 ft. by 2 ft. 6 in. These four and a-half plugs of gelatine will occupy a height of 2 ft. 6 in. when rammed home, leaving a burden of 4 ft. 6 in. on the top of the charge, and only 2 ft. 6 in. from the hole to the face of the stope ; what is more likely than the charge breaking through into the stope, leaving 3 ft. or 4 ft. of the collar standing, and if the outside holes fail to bring their burden, it is certain the inside ones will not bring theirs, and so the whole round is lost. I have seen this occur, and so I prefer to drill my holes 6 ft. deep with a burden of 3 ft., and give them an extra half plug if necessary. The placing of the holes in stopes, as described by the author, I have seen carried out on our mines by old miners in the '80's, and from my experience of our old machine men and hand stopers, extending over twenty years, I am convinced, that, what they do not know about placing holes, is not worth knowing. I am sorry that the author was not able to finish his experiments on the new cut, but I can tell him he will not find it a success in a solid face. This cut if applied to a solid face is certainly a new one, but if used for taking out a cut to a slip is as old as the hills. Drilling both over and under the arm in stoping is a general practice with the old miners here. It is the younger ones who have not got into the way of it yet. I fear that the majority of non-mining men on these fields have an idea that they can always turn their hand to mining when everything else has failed, but my experience of these men has been that they would make a failure of anything they tried. They certainly are not successful as miners, and I maintain that these are the men who, having succeeded in obtaining a blasting certificate, have lowered the average efficiency of our miners. The indiscriminate way in which blasting certificates are issued here is the main factor of the low efficiency of our miners as compared with other countries. No man should be given a temporary certificate even until he has worked three months under a thoroughly practical miner, and no permanent certificate should be granted until the man has served six months underground, and yet we have men here running machines and holding blasting certificates after being underground only two or three weeks. I have noticed one great fault on some of our mines here, and that is not allowing the machine man an adequate supply of drills. To enable a man to get through the maximum amount of drilling he ought to be supplied with at least four good drills for every hole he is expected to drill ; if he is so provided, he will be able to drill six holes with each machine instead of four, which is the average number at present. The machine fitter should have his shop underground instead of on surface, and should keep a plentiful supply of spares, and also complete machines. Many a shift's drilling has been lost through a machine breaking down at the commencement of the shift, no other being obtainable until noon or at the end of the shift. Drills, if not sharpened underground, should be sent below between shifts, and not after the men have gone down ; and, as far as possible, the heads of all bolts on a machine should be of uniform size, so as to need only one spanner.

Labour.—More white labour might with advantage be employed here by splitting up the big gangs of boys, as no man can economically boss up a gang of boys over 25 to 30 in number unless they are all in one stope, in which case a man can boss up his drilling boys and also his "lashers." Prior to the war, in the majority of cases, a man had to boss up his own hammer, shovel and tram boys and do all his stope timbering, and this, I maintain, a man can do at the present time. Under our present system a man running hammer boys has nothing whatever to do after he has started his boys drilling, until it is time to make up the charges preparatory to blasting ; during the interval, a matter of five hours, all he has to do is to sit on a soft plank and watch the boys drilling. I have no time for white labour on anything but skilled or semi-skilled work, and have never seen a white man yet who could compete with a Kafir for shovelling or tramming and *keep it up*. The experiment has been tried over and over again and always ended in failure. If work must be found for our unskilled white unemployed then it must be found on the surface, as there is no room for them underground. I am sorry to hear that the author has found that the great majority of the worst men we have are among the old miners, and it would be interesting to learn how he has arrived at that conclusion. Has he ever tried working old miners under the supervision of *practical* mine captains and shift bosses, for nothing takes the heart out of an old miner more than being bossed up by one who is fresh from some technical college and whose practical knowledge consists of six months' sampling and surveying. The suggestion that every manager should be compelled to visit the other mines of the group is a good one, but not as good as the compulsory holiday each year, presumably on full pay ; but why not extend the practice to the mine captains and shift bosses, as I am sure they would appreciate the holiday, even if they did not benefit by the "free education" offered by visiting the other mines.

Mr. C. O. Schmitt (*Member*) : The author, in his highly interesting paper, has touched upon many features of mining on the Rand, and while I am unable to criticise the greater portion of his remarks, I wish to deal with a few points raised in the paper.

He is, no doubt, right when he states that the principal saving in working costs must be looked for in underground operations, seeing that by far the greater portion of working expenses are accounted for by work done below the surface. It is admittedly impossible to fix a proportion between working expenses as due to mining operations and reduction works costs, since the former are a variable quantity, not only as between mine and mine, but also in the same mine due to changes in the size, position and hardness of the ore bodies. Still, when considering working costs, it must not be forgotten that in one mine at least, the whole reduction works expenses are only 3s. 6d. per ton milled and treated. While this figure cannot necessarily be attained in every mine, it will show that in many mines there is yet room for improvement on the surface.

In discussing the question of hauling the ore to the surface, the author begins by saying that there are too many shafts and engines in use, forgetting that shafts and their equipment are the result of a slow but constant growth, and if the engineers responsible for the design of these plants could see them in the light of the present day they would be the first to admit the absurdity of some of the arrangements, such as four-compartment shafts, to mention only one item. On the other hand, it is by no means certain that the author and, for that matter, many of those present would have done what is now considered the best practice. Most likely, history will repeat itself, and if the author should perpetuate his work for a future generation to criticise he will in all probability be severely taken to task for not doing the right thing.

If he will only look around, he will find a good many deep level mines with one or more shafts abandoned long before they came within reach of the reef. No doubt, a lot of money was spent on these shafts, but still more, which might have been spent, was saved for other purposes, reduction works, for instance, thus anticipating the author's ideas.

It must be admitted that the author not only criticises, but he also puts forward suggestions, one of which is the proposal to hoist 5,000 tons of rock per day through a two-compartment shaft, or rather a three-compartment shaft, having two hauling ways and one pump way. I felt very much inclined to regard this as a misprint, but as the figure mentioned recurs in various places we must take the statement seriously. It is most unfortunate that he did not work out the problem for the benefit of those who might be sceptical. He seems to be doubtful as to the size of his ideal shaft, because in one place he says that the size of the shaft should be determined by considerations of ventilation only, while somewhere else he repents and states that the three-compartment shaft, which he recommends, would be too small for ventilation purposes and, while explaining the wonderful feats performed in winding in other parts of the world, he entirely forgets to tell us how big a shaft is really needed to provide for the proper natural ventilation of the workings which supply the 5,000 tons of rock per day. Let us hope he will make good this deficiency.

It appears to the writer that if a shaft is large enough for hoisting purposes it can certainly be made to suffice for the ventilating requirements. No one would dream of sinking a five-compartment shaft to a depth of 3,000 ft. or more, if three compartments could deal with the rock, men, tools and stores. The saving between the two alternatives would pay for the installation of an efficient ventilating plant with a handsome margin to help to pay for development or the reduction works. This ventilating plant need not necessarily be installed at the surface, but might with advantage be located at the bottom of the down-cast shaft (since there must be two exits from a mine) from where the air could be distributed to those bad places in the mines, spoken of by Dr. Moir. This would make certain of the fresh air reaching the points where it is most needed, and would avoid its passing by the nearest route from the down-cast to the up-cast shaft.

A good deal of money is at present spent upon ventilation by means of the available compressors blowing air into the working places. The amount of air so delivered is very small, and it appears that the power thus used during a certain period every day would in itself pay for the continuous running of an efficient ventilating plant and with a better result. However, on mines where ventilating by compressors has to be practised, a considerable saving in coal can readily be effected by converting the usual two-stage compressors temporarily into single stage machines so that both high and low pressure cylinders deliver air directly into the mains. This would, in most cases, be equivalent to adding 50% to the volume of air delivered, or, if only a certain quantity of air be needed the coal consumption due to ventilation only could be reduced by 33%. It might also be claimed that the air would be less contaminated with oil, since it passes through one cylinder only. For all I know, the above method may already be in use in some of the mines on these fields, and if so, it might be interesting to hear how it satisfies the requirements, or whether it fails to do so.

Returning to the author's proposition of hoisting 5,000 tons of rock per day from a depth of 3,000 ft. through a shaft having two hauling ways, and leaving sufficient margin for handling men and tools, it will be interesting to see how the problem works out. I take it that 20 hours would be the maximum running time, and leaving out the question of men and tools for the moment, there would have to be hoisted, say, 240 tons per hour, or 4 tons per minute. With a 4-ton load there would have to be 60 trips per hour, or 60 seconds for the trip, including stoppages. Allowing 10 seconds for loading the skip below and discharging the one in the head frame, which allowance is by no means great, and also allowing 20 seconds for acceleration and 10 seconds for the retardation period, there remain 20 seconds for the constant speed run. The maximum speed would, therefore, be approximately

$$\frac{20x}{2} + 20x + \frac{10x}{2} = 3{,}000$$

$$\text{whence } x = 85 \text{ ft. per sec.}$$
$$= 5{,}100 \text{ ft. per min.}$$

or practically a mile per minute. The mean speed would be 3,600 ft. per minute. It will be admitted that this is somewhat fast, requiring a very straight shaft with heavy steel guides, conditions which will be found in very few mines.

To reduce this speed it will be necessary to hoist a greater load in each trip, and assuming a 6-ton load we have 90 seconds available for the whole trip. Making time allowances as before, the constant speed run will extend over 50 seconds, and the maximum speed will be 46 ft. per second or 2,800 ft. per minute. However, 20 seconds for the acceleration period is too small for the heavy loads to be dealt with, requiring an engine of unreasonable dimensions. Making allowance for 30, 40, or even 50 seconds for acceleration, the maximum speeds would be 3,000, 3,280 and 3,600 ft. per minute respectively. The latter figure seems quite high enough for the continuous and regular work which is required on these fields. If it is remembered that the rope required would have to be 1½ in. diameter, and of good quality it will be seen that a drum of at least 14 ft. diameter is required again needing an engine of long stroke or large cylinders. In any case, the engine will be very heavy. Still, in accordance with the above, it must be said that it is possible to hoist 5,000 tons of rock per day through two hauling ways, although there is no time for dealing with the men.

To break the specified quantity of rock in two shifts there will be required, say, 800 white men and 7,200 boys working in two shifts or, say, 4,000 per shift. It is, of course, highly necessary to send the men below expeditiously so as to avoid too much overlapping of shifts. Assuming that the load to be put on the rope when hauling men is only 90% of the rock load, skip and cage being of the same weight, 70 men could be handled per trip or, say, 60 trips for lowering a complete shift. The time required for a trip will not be far from two minutes, so that it may be safely assumed that two hours are needed for the lowering of one shift only. Assuming that the afternoon shift is lowered when the morning shift comes up, the total time required for handling men would be six hours per day. In addition to the men, drills, timber, explosives and other stores have to be sent down, which will account for at least one and a-half to two hours of the winder's time every day, so that men and tools account for practically eight hours per day.

It is obvious, therefore, that a shaft, as suggested by the author, could not handle the rock as well as the men, thus requiring a further two compartments with a separate winder at least as big as the rock hoist. Even a five-compartment shaft will probably be thought too small for the specified work, and I dare say, no responsible engineer would recommend anything less than seven compartments for a similar proposition. With a shaft of seven compartments the work could be conveniently done, while with a five-compartment shaft there would only just be the bare time available for maintaining engines, hauling ways and loading boxes in proper repair. The author maintains that the shaft timbering is not needed in many cases, and I trust, for the sake of saving capital outlay, that he is right. However, even if one does not go as far as omitting the timber altogether, a good deal of timber could be saved by using substantial steel guides and putting the setts 8 ft. centres instead of at 6 ft. or less. Steel guides should be of T section, similar to those put in at the Simmer Deep (Fig. I.), for the reason that this type of guide gives plenty of metal for wear, promising a compara-

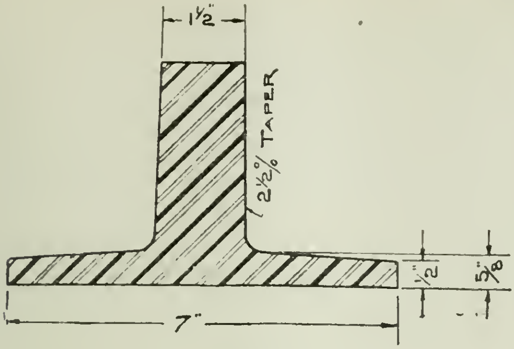


FIG. I.

tively long life. An endeavour should be made to reduce the wear on the guides by using rollers on skips and cages instead of shoes as at present. Such rollers have been in use at the Robinson Deep for some time with conspicuous success. Steel guides of channel section (Fig. II) do not seem to give the same life against wear as those of T section, although they are stronger in the axis parallel to the base, thus rendering the shaft timbering more rigid. The weight of either type need not exceed 30 lb. per linear foot.

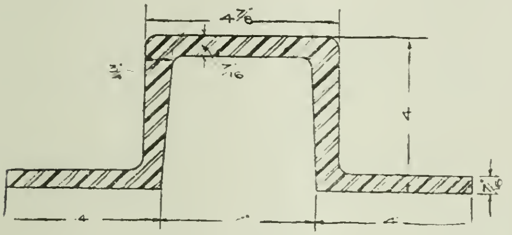


FIG. II.

Referring to compound shafts, the author condemns this type outright, but it should be said that, while there are many valid objections to them, there are also, in certain cases, advantages to be derived, which justify their adoption. Stage winding is by no means the unmixed blessing it might appear at first sight. It seems unreasonable, and can be shown to be so, to install a separate winder at the bottom of a comparatively shallow vertical shaft if the boundary of the property can be reached conveniently in one stage. No doubt, where the underground inclines are very flat and will allow of using the truck system of haulage, which is by no means general on the Rand, there is no justification for a compound shaft, no matter how shallow the vertical shaft. However, inclines on the Rand are, as a rule, over 30°, so that the general adoption of the truck system of haulage does not appear to have much chance.

The trouble generally experienced with compound shafts is mostly due to avoidable causes. A properly designed curve and suitable skips will considerably help to ensure smooth working. Why skips of compound shafts, where about two-thirds of the entire trip is on the incline, should be provided with the miniature wheels one generally sees is past understanding. If large back wheels on skips are needed for incline shafts pure and simple, they are certainly also needed for skips in compound shafts, unless the length of the incline be negligible.

The author starts off by saying that there is at present too much machinery in use on the mines, but it seems that by indiscriminately discarding compound shafts he re-introduces the very defect he wishes to rectify. The advantages of compound shafts can be best shown by considering the case of a mine where the combined depth of vertical and incline shafts does not exceed 3,500 ft. of which the vertical is, say, 1,000 ft. deep, and the angle of incline, say, 30°. The amount of rock to be hoisted to be 2,400 tons per day, or 120 tons per hour which, with a 4-ton load, gives 30 trips per hour, and allows 120 seconds for the complete trip, including stoppages of, say, 10 seconds at the end of each trip for loading and discharging. Allowing further 20 seconds for acceleration and 10 seconds for retardation, the maximum hoisting speed will be

$$\frac{20x}{2}+80x+\frac{10x}{2}=3{,}500$$

$x = 37$ ft. per sec., or 2,200 ft. per. min., the average being about 1,900 ft. per minute. Allowance should be made for a slightly decreased speed when going round the curve so that the maximum speed would be somewhat higher, but against this must be set the fact that winding does not begin at the lower boundary of the property, and that the extreme conditions assumed above only prevail towards the end of the life of the mine, average conditions being very much more favourable.

Now, considering the same case, but with separate engines for vertical and incline we shall find that, with the exception of the drums the engine required for the vertical alone will not be much smaller than that for the compound shaft for the reason that, even if the load per trip be decreased the time lost in extra stoppage due to the more frequent trips, will largely balance the gain thus obtained. It should also be mentioned that starting conditions in a compound shaft are more favourable than for either a vertical or an incline shaft if considered separately.

With regard to the incline shaft, a 3-ton load might perhaps be used, requiring 40 trips per hour, and giving a maximum speed of approximately 2,300 ft. per minute. The engine for this would be a comparatively small one, but its installation underground with arrangements for power supply, cutting station, and bins, providing loading boxes, skips and dumping arrangements would mean a considerable expenditure. The difference in the cost of the two winding plants may be taken to be anywhere from £15,000 to £16,000 which, in these days, is quite a useful sum for investing, either in development or in the reduction plant.

In cases where the depth of the vertical shaft is 2,000 ft. or more, with a corresponding increase of the total depth, stage winding will have to be adopted, but even in this instance the winder for the incline should be kept on the surface so that the steam plant supplying the winder for the vertical shaft can also supply the former. The ropes will have to be run down the vertical shaft, which means an extra pithead pulley for each rope. This arrangement will give several advantages, viz., a smaller boiler plant than if the power were sent to a winder below by either compressed air or electricity and, since both engines are in the same room, excessive peak loads can be avoided by omitting to start both winders at the same time. There will also be a saving in capital expenditure. This system of winding has been in use on the Rand for many years, and in one case with a vertical shaft of 2,500 ft. depth. The limit of its applicability might be said to be about 5,500 ft. combined depth, depending on the depth of the vertical shaft.

I have previously admitted that the use of truck haulage on the inclines underground would at once do away with compound shafts but, unfortunately, the steep angle of the reef precludes in most cases the use of this system. Haulages on an angle of 20° and even 25° may be quite satisfactory, but to seriously propose them for angles of dip up to 45° seem open to objection. I do not remember the maximum angle on which truck haulages are in use in other countries, but perhaps the author can enlighten us on this point. There should be plenty of room for testing this system on the far East Rand. Putting boilers underground, when transmission of power has reached a high state of perfection and efficiency, seems also a retrograde step. There is, I take it, some excuse for it in coal mines, but certainly not here, where coal and probably water would have to be sent down. The capital outlay, especially on cutting the large station, required would be enormous, not to mention what would happen should bad ground be met with.

I wish, in concluding, to apologise to the meeting for having at such length dealt with only one of the many points raised by the author, especially since many members probably care more for ways and means for dealing with the ore once on the surface than for the mere detail of bringing it from below.

Mr. A. Richardson (*Member of Council*): That there is more scope for improvement below than above ground is an opinion which engineers on the Rand have held for many years, and that it has been and is being given effect to, the steady reduction in mining costs clearly shows; but that we have to travel a long way yet before we can take our place in the front rank as mining people is a statement which requires a much stronger argument than the absence of record hoisting performances to prove.

A community which has developed model methods of mine sampling, valuation, and surveying, which holds the world's records for rapid shaft sinking and underground development, which has converted a stretch of barren veld into the premier goldfield of the modern world is entitled to consider itself in the front rank, or thereabouts, of mining people. In this connection let me quote the words of Mr. J. T. Stobbs, a colliery engineer, in a recent mining discussion before the "North Staffordshire Institute of Mining and Mechanical Engineers." He said, "there was no doubt whatever that so far as modern mining practice was concerned, if they wanted to see it at the best they must go to the Rand for it." If it were necessary for us to hoist 5,000 tons through two compartments in 24 hours it would assuredly be done, however heavy the expense incurred, but as the 1,000 stamp mills freely spoken of during the past ten years seem to be still in the same stage of construction the problem cannot be described as pressing; it is interesting, however, to speculate on what amount of rock could be hoisted if we, so to speak, hustled. This amount will depend on quick loading from large and perfectly equipped boxes serving several levels and fast winding with powerful engines, well maintained shafts, and expert drivers. Making the bold assumption that the mechanical arrangements are highly efficient the question then becomes one of winding speed. As far back as 1878 the California Gold and Silver Mining Co. was hoisting from a depth of 2,500 ft. in 45 seconds, an average speed of 3,333 ft. per minute; the No. 3 Tamarack has done 4,800 ft. in 1 minute 15 seconds, or 3,880 ft. per minute, the maximum speed during the wind being over 5,000 ft. per minute; this hoist has made 40 trips in 1 hour with 3 tons of ore per trip; the Calumet and Hecla has wound from 4,900 ft. at an average speed of 3,500 ft. per minute.

At Kimberley 4,000 short tons have on several occasions been hoisted in a little over 11 hours from 1,560 ft.; with this engine the acceleration period occupies 16 seconds, the retardation period

13 seconds, the maximum and constant speed period 13 seconds, equivalent to 3,770 ft. per minute, the average speed being 2,230 ft. per minute for a winding period of 42 seconds; the loading and dumping together occupy 5 seconds, so that each trip takes 47 seconds. Taking some of the leading British collieries, which show fast work under normal conditions, the following represent good practice:—

Dowlais Cardiff, 2,220 ft. in 52 seconds, an average winding speed of 2,562 ft. per minute; Ashton Moss, 2,850 ft. in 1 minute 25 seconds, or 2,010 ft. per minute; Lady Windsor 1,500 ft. in 35 seconds, or 2,571 ft. per minute; Bolsover, 1,116 ft. in 28 seconds, or 2,388 ft. per minute; Denaby and Cadeby, 2,289 ft. in 55 seconds, or 2,497 ft. per minute; Rhodes Rotherham, 1,650 ft. in 45 seconds, or 2,358 ft. per minute; while at the Rosebridge colliery a maximum hoisting speed of 5,100 ft. per minute has been recorded; such an excessive hoisting speed, however, as 5,100 ft. per minute does not represent sound work, this, for considerations of safety, should not exceed 4,000 ft. per minute. From the above, and in view of regularity of output, an average hoisting speed of 3,000 ft. per minute may be assumed to represent the best modern practice.

Now the time in seconds of each complete wind is : $\frac{\text{depth of shaft in feet}}{\text{speed of winding in feet per second}}$ + number of seconds occupied in loading and dumping. Assuming the last term to be 10 seconds, the time of each complete wind from an average depth of 3,000 ft. will then be 70 seconds, giving 51·4 winds per hour, or an hourly output of 205·6 tons with 4-ton skips. If this is maintained for 20 hours out of the 24, a very excellent performance, the output will be 4,112 tons. With 5-ton skips this figure would be 5,140 tons. This estimate is on a very favourable basis, and outputs of such magnitude would be considered phenomenal in any part of the world. In South Wales, for instance, an output of 1,000 tons through one shaft in a daily shift of 10 hours is considered good work, and 1,500 tons exceptionally so, and this from depths which in only one or two instances exceed 2,000 ft.

With regard to the balancing of the load, there is no doubt that a tailrope is the most perfect method known, as conical drums will only completely balance when winding from one fixed depth, but unfortunately the use of these balance ropes is attended with many disadvantages. With them independent trips in either compartment are impossible, and the changing of levels leads to much loss of time; the sump also has to be extended to accommodate the sheave and slide, and there is much extra weight thrown on to the capping, and also on the safety catches should occasion ever arise for them to act; it is a dangerous device to instal at depths greater than 2,500 ft., because the swaying or whipping action of the rope tends to destroy the guides and timbering. The shape of shafts has been a much debated point ever since shafts were first sunk, and as far as I am aware there has never been any authoritative pronouncement on the subject, and one is very badly wanted; apart from any special considerations it would almost appear as if, in this matter, engineers all the world over followed the very sensible advice given by the old proverb to those about to visit Rome. The circular shape seems to offer no advantages to us here. Where, as in collieries, ventilation is of vital importance on account of explosive gasses the waste space in circular shafts is an advantage, and where the nature of the strata necessitates cast iron tubbing to resist the heavy pressure it is difficult to see how for engineering reasons any shape other than circular could be adopted. In Wales, oval as well as circular shafts are seen, and in Scotland and America the usual shape for colliery shafts is rectangular.

With regard to guides, the general opinion at home is that steel rails are the safest for fast winding from deep shafts with heavy loads. It is worth noting that in Germany the winding of men on rope guides is prohibited; their use also necessitates a skip which cannot turn over but must discharge through hinged doors, and a bend is out of the question. Now, bends are mighty convenient things, and do not possess half the terrors with which the author so generously endows them. They save stage winding, and, therefore, that duplication of plant which the author so deplores on the surface, and which is even more to be deplored below, and that they are quite easy to get men through a trip down the Wit Deep will comfortably prove. Taking mines like the Jumpers Deep and Geldenhuis Deep, which turn their shafts off at moderate depths it is difficult to see where stage winding would come in.

As to the speed at which men are hoisted in shafts the following recent statement of Mr. W. H. Pickering is interesting:—"Already there are instances where 80 men ride in the shaft at one time, and at a speed of 40 miles an hour." This means a speed of 3,520 ft. per minute, or 59 ft. per second. And with reference to the rarity of accidents in shafts, it may be pointed out that in 1907 there were 71 accidents in British collieries, causing the death of 97 persons, and that in this 71 were not included those shaft accidents which fortunately were not attended by loss of life.

The idea of putting boilers down below is scarcely practicable. The danger of fire would call for steel "timbering," an expensive item, and on steel the gases of combustion plus acid water would exercise a serious corrosive effect; the excavation of a chamber to accommodate the boilers and bunkers would prove a costly piece of work, and the difficulties of supervision and repair would add considerably to the running costs of such a plant, moreover there would be the danger of asphyxiating a number of men should the cage be hung up in the shaft. This danger is recognised in certain collieries by an arrangement whereby communication can be made from the cage to the onsetter and fresh air admitted to the shaft. There are, however, as the author says, several home collieries where boilers have been installed below: Wearmouth, Ashton Moss and Clifton Hall, for instance, but in the last-named no winding whatever takes place in the upcast shaft, and as these shafts have barely adequate furnace ventilation the addition of a few boiler fires is a great help. Endless rope haulage is usually confined to moderate inclines though there are, I believe, one or two working at over 30°; this system would be less simple and elastic for shaft work than the ordinary hoisting method and much more liable to breakdowns, still for flat dipping properties like the Geduld it might prove very serviceable. To prevent slight defects leading to big smashes where the gradient is heavy, the road would have to be well lit, inspected, and provided with switches at suitable intervals, so that the haulage motor could be stopped in the least possible time; the under rope system would be preferable and safety catches, of which there are now many types, both for full trucks coming up and empties going down, would have to be put in every 30 ft. or so to prevent runaways; to economise timber the rails could be laid on longitudinal concrete sleepers as at the Angelo Deep.

The sinking of two shafts close together would involve elaborate precautions to avoid the short-circuiting of the air current through the stopes from one shaft to the other: these measures are not difficult to carry out in collieries where the workings are laid out on a definite plan, and where extraction proceeds with symmetrical regularity from year to year, but they present quite another problem when applied to steep dipping gold mines with workings on two or three reefs whose fluctuating gold contents necessitate a more or less erratic manner of advance; again two shafts at some distance apart would provide better facilities for judging of the average value of the reefs and the nature of the ground than two close together could. It is estimated that the substitution of two small shafts close together for one large one would entail an additional expenditure of 30%.

The hanging wall of a stope in whose foot wall it is intended to trench the shaft will, if it is affected by subsidence, cause more trouble, delay, and expense, by closing up the shaft than most mining men care to invite; the better way would be to sink the shaft in the solid country at least 30 ft. below the reef series.

The idea of specialising in drilling and blasting has been adopted in many mines, those using the hole-contract system, for instance. The North Star mine of California has adopted it, and as we have heard a good deal about "Californian efficiency," it might be pertinent to enquire if this is one of the factors on which that efficiency is based; also at the Daly West mine, where two 8-hour shifts are worked, special powdermen go round each shift, measure the holes, and then distribute what they consider to be the correct number of cartridges for each hole with cut and capped fuse; it is claimed that by this means the consumption of 40% dynamite has dropped from 1—1½ to ¾ lb. per ton broken. Grooved steel, costing 2⅜d. per lb., gives very good service both for hand and machine jumpers, and for the reasons given by the author is preferable to the dearer brands, moreover, where hand jumpers are sharpened by a machine which bites off a bit every time the use of a good quality of steel is not economical. The statement that subsidence will travel to the surface with little diminution is quite at variance with experiences on the Rand and elsewhere, as was clearly shown in my paper on the subject and the discussion thereon, not only in some of these cases did the subsidence not reach the surface at all, but in all those in which it did the amount was very sensibly diminished.

To keep the stopes with large amounts of rock in them is rarely possible with large mills and narrow reefs, nor would it be advisable to do so if, as the author suggests, no pillars were left, only ribs along the level; with backs of 200 or 300 ft. the result would probably be that much of the hanging wall would repose on the broken rock and very likely on the shovel boys, too.

The three-hole parallel cut is said to be a useful one in soft but a poor one in hard rock. The lowest hole should look down a little to permit of its being half filled with water so as to prevent it closing up at the bottom when the top hole is fired.

The suggestions regarding holidays will heartily commend themselves to all managers and heads of departments, to whom it will be rarely necessary to apply force to ensure their acceptance of an offer of a few weeks at the coast, but the encouragement to undertake the expensive journeys that visits to other fields would entail might have to be of a financial nature.

In conclusion, I congratulate the author on a paper that must have involved a great amount of work, the result of which is so stimulating.

The President: As Mr. Johnson will not return before August, although this is really the last time for discussing this paper, I suggest that the paper be held over for further discussion.

RESCUE APPLIANCES: LESSONS FROM GLENCOE.

(*Read at March Meeting, 1908*).

By H. Kestner.

DISCUSSION.

Mr. Tom Johnson (*Member*): I must take exception to the author's statement that the Shamrock type of apparatus was exclusively employed by the rescue party. The German party under Mr. G. A. Meyer used them, but I fancy the Paris firemen used the Dräger also; after the German rescue party left, the Pneumatogen II. was used.

The author does not seem to have much use for smoke helmets, yet it is a fact that smoke helmets have been the means of many lives being saved, whilst the more elaborate apparatus has not yet any lives to its credit. As I have already mentioned, smoke helmets, etc., would be very useful in cases of gassing.

The Shamrock is a good apparatus, so are the Fluess-Siebe Gorman, Aerolith, Dräger, Weg, and Pneumatogen in their latest forms.

I should certainly recommend that the first rescue station should have all types, so as to see which suited best, and that the Government, in conjunction with mine owners and Fire Assurance Societies, should go into the matter. The first central station could be in connection with a fire station, firemen, in my opinion, being men who are at present under a discipline that is very necessary in this kind of work, and miners from the different mines could attend the station for training. Also the mining students of the Transvaal University College should go through the training as part of their work, for although many of them would not be fit for the actual rescue work with the apparatus, still a knowledge of the apparatus would be very useful to them in their after careers. For anyone wishing to follow up what has been done in regard to rescue apparatus I send a list of papers,* etc., which have been written on the subject.

A FEW NOTES ON STAMP MILLING.

(*Read at April Meeting, 1908.*)

By W. H. Jane (Associate).

DISCUSSION.

The President: Referring to the first paragraph of the author's paper, I think his view of the cycle theory of five stamps is quite correct. With regard to the setting of cams in a certain order of drop, it is still a very open question among millmen as to which is the most advantageous setting, some insisting upon one certain order, others obtaining as good results with an order which would be condemned by the first party, and so on. Personally, I have tried many orders of drop, and must say that I prefer the order 1, 4, 2, 5, 3, counting from whichever end of the shaft one may choose. I run with as full a drop as possible at 101 to 102 drops per minute, and do not find any trouble with packing of the ore at the ends of the mortar box. There may be others who will condemn this order of drop, but I only state my experience. If one reflects a moment, it is but reasonable to assume that the end stamps in a box might require a little longer drop than the others, as stamps 2, 3, 4, I will call them, have the advantage that in dropping they are able to distribute what ore may be disturbed by their impact upon the die, partly under and partly round the front and back of their neighbours, whereas the end stamps, 1 and 5, have only one side open for such distribution, as the ore thrown out by them on their other sides must be thrown against the ends and corners of the box, whence its only escape is to wash under the stamp when lifting. However, as before stated, I do not find any trouble with banking at the ends of the box.

With regard to the number of drops per minute, this is a point which I consider very largely depends within limits upon the character and size of the ore fed to the stamps. A soft or a finely broken hard ore can be advantageously crushed with more rapid blows with a shorter drop than an ore which is hard and is supplied to the stamps in large lumps which require a heavy blow to break them up. Here again I would revert to the great necessity of making the crusher station what it really should be, so as to supply an as even sized product as possible to the stamps. With regard to waste space in the mortar boxes I think a distinct advance has been made in this respect in the straight back, round cornered boxes, now made by some of the leading firms of manufacturers.

Regarding feeder drives, this is a subject upon which I hardly dare to touch, for fear of being dubbed a "crank." There has been much controversy upon this point, and I would only call the attention of members to the very simple device laid before the Society by me and described in our *Journal.*[*] The original device then shown is still in use and appears good for a long time yet, it seems to take up its own wear and gives no trouble whatever, and owing to the satisfaction it has given I have installed several in the mills under my control. I think this device would meet with the author's approval in having no springs to pull back, it doing this by virtue of its own weight, and the balance of the rock shaft can be very easily regulated.

---

* See Notices and Abstracts, this *Journal*, p. 35.

With regard to the relation of tube mills to stamps, and the percentage of work done by each, we are, I think, still in the experimental stage, as the size and shape of tube and speed of revolutions of same has not yet been definitely settled, and the whole matter greatly depends upon the size of the screening in use in the stamp mill, and also upon the degree of fineness to which it is desired to reduce the final product.

NEW TYPE OF NATIVE COMPOUND BUILDING OF ALL METALLIC CONSTRUCTION.

(*Read at April Meeting, 1908.*)

By C. B. Kingston, B.A., B.A.Sc. (Member of Council).

DISCUSSION.

Mr. C. O. Schmitt (*Member*): The author of this paper is to be complimented on bringing forward a novelty in the way of compound construction, although I cannot agree with him regarding the merits of the design as a human habitation. Buildings of this type have been erected all over the world as warehouses and for similar purposes, but never, as far as the writer is aware, as living quarters. Cheapness is the undoubted feature of this design, but this by no means exhausts the considerations to be borne in mind when housing the unskilled workers of this country.

The author deliberately tells us that this compound has been tried in hot weather and in cold weather, and while I have no doubt that it withstands heavy storms of rain and wind, I would like to have a little more information with regard to the inside temperature on a hot summer afternoon as well as during a cold winter night. I have not yet heard of anyone claiming architectural beauty for the ordinary compound, so that the author's disclaimer for his design can only be taken as a jocular remark. However, to seriously tell us that the structure appealed to the native eye by way of similarity to his grass hut is libelling the native's sense of discrimination. Surely, the grass hut, which is cool in summer and warm in winter, is a much superior dwelling, at any rate, from the natives' point of view. It has always been understood that a dwelling, whether for whites or blacks, should without artificial heating maintain, as near as possible, an even temperature throughout the year. It would be interesting to hear how the author's model building fulfils this condition.

Considering the details of the structure, I find from the sketches submitted that the area of the windows cannot exceed 3% of the floor area, so that the Municipal bye-laws would have prohibited the erection in its present form, had such been attempted within the boundaries of this town. The use of 24 gauge galvanised iron for a self-supporting roof is also open to objection for the reason that the iron, which is likely to have its protecting coat damaged during the rolling process will have a very much shortened life compared with similar iron merely nailed to the wood frame of a roof. In Europe 22 or even 20 gauge iron is considered essential for this purpose. Further, 24 gauge iron will not allow of a good riveted joint. I believe the author will be disappointed in his estimated life, because the fumes from the open fires, which the boys will, in spite of regulation stoves, maintain on the inside, considerably aid the atmosphere in its deteriorating action on the outside

The author refers to the maintenance of brick-walls in the present type of compound, but there is no need for such maintenance, provided the walls are built of well burnt clay bricks, and not of an imitation consisting of red soil and ashes. Good bricks can be had for the same price as bad ones, and cartage and labour in building is the same. The same may be said of lime and cement.

With regard to the cost of rooms of the author's type, he states that 18 rooms averaged £2 12s. per boy, and a little further on gives the cost of a complete compound as from £2 17s. 6d. to £3 15s. This is evidently a slip, because I cannot conceive how the author can provide the remainder of the compound from 5s. 6d. to £1 3s. per head, which usually consists of kitchen with boiler, change house, fresh water service, fire service, lighting, time office, and fencing, and which cost from £4 to £5 per head. This, added to the cost of living quarters, brings the total to from £6 12s. to £7 12s.

I admit that some of the existing compounds, especially those erected for the Chinese Coolies, to which the author evidently refers, when he speaks of a total cost of from £7 10s. to £10 are perhaps more elaborately built than would be done for natives, although I always understood that comfortable housing tends to improve the natives' efficiency.

---

[*] See this *Journal*, Nov., 1907, p. 147.

Brick compound rooms, 33 ft. 6 in. square, each room to house 75 boys and allowing 240 cub. ft. of air space, or 20% above the requirements, can and have been built for £4 per head, allowing a fire-wall for every three rooms. Adding to this £4 or £5, we have a total cost of from £8 to £9, as against from £6 12s. to £7 12s. The difference in favour of the author's type of room of £1 8s. per head could, by decreasing the air space to the regulation 200 cub. ft. and bringing the window area to 3% of the floor area, be reduced to about 16s. only, which item it is safe to ignore when dealing with the health and comfort of the necessary native worker. From a sanitary point of view, there is nothing to be said in favour of either construction, because simple whitewashing of the walls will do all that is required for the brick construction, and the cost of fumigating is the same for both.

Concluding, I would like to remind the author that the extra 16s. per head spent on a brick compound, if a loss to the company, which in view of the extra comfort cannot be admitted, will be the gain of the country, because all materials with the exceptions of corrugated iron, bunk framing and glass can be and actually are produced in this country. The mining industry might do worse things than buy locally made bricks, cement, lime, tar, door and window frames, and use local timber for the framing of the roof. This means that all but 20% of the money spent will remain in this country.

THE ADAIR-USHER PROCESS.

(*Read at May Meeting, 1908.*)

By Alfred Adair (Member.)

DISCUSSION.

Mr. S. H. Pearce (*Past President*): Our Past President (Prof. Yates), in commenting on this paper, congratulated Messrs. Adair and Usher for their persistence and courage in developing the process that bears their name, and I should like to add that while it is not my intention to say much about the process at this time, I feel it is only due to the author to state how much I appreciate the volume of accomplished work that is shown in the paper he has submitted. I may not be an enthusiast in regard to the beneficial results obtained from the use of umber, which is due to the fact that, as far as my experience goes, our ordinary method of treating current slimes is in itself sufficiently self-aerating and oxidising as to render the use of oxidising reagents superfluous. Further, as I have had no experience with this reagent, I will leave it to others, who have, to give their opinion.

As regards the Usher washing apparatus, it is now pretty well known that the washing of slimes in suspension was carried out by a G. S. Duncan, who holds patents in this country as far back as 1899, for a certain form of plant. The apparatus now in use here is a modification and simplification of that idea, worked out so as to be adapted to existing plants, and has been proved with beneficial results, particularly in the case of those where capacity has been taxed beyond the normal limits, and in that I can join with Prof. Yates in his congratulations, which I hope to augment when I have had more experience with this mode of treatment than I have at present.

I may say, however, that I am not at one with Prof. Yates' remark as to our ever being so conservative as to consider the last word has been said about cyanidation, at any time, either in the past, the present, or, for that matter, so long as we have brains to think, and the funds to experiment with. I think I might add that there are very few here connected with the cyanide process, who have not some ideas of their own on the subject.

The author has brought forward another modification of the washing process, which aims at continuity, and is deserving of every consideration. Unfortunately, as he himself states, it involves a considerable change in our ideas of plant, and without former experience in this respect we have to make haste slowly. In at least one respect I do not share his optimism, and that is in regard to the cost of reducing battery pulp by tube milling to fine sands. Since the publishing of that paper by Mr. Caldecott and myself as to the estimation of tube mill work, we have had good reason for appreciating the enormous amount of energy consumed in reducing fine sands but very little smaller, and which, on referring to the tables in that paper, will be seen to absorb the lion's share of the work units.

He assumes that 60% of the pulp can be reduced to 90 mesh for 5d. per ton, and therefore the whole should be 7½d., but might be 1s. per ton. He overlooks the fact that of this 60% at least 35% has already been produced in the battery. Further, I regret to say that we have not yet reduced the cost of tube milling so low as 5d. per ton milled, however, optimistic we may be as to the future. I do not propose to give any figures here, but I may state that I have gone very carefully into the pros. and cons. of finer grinding, and am of opinion that judging by present practice and knowledge it is premature to discuss the all-sliming question. One of the considerations, apart from the cost of grinding, is the occurrence of the gold in conjunction with the pyrites, which had to be ground to an extremely fine state of division to yield the precious metal with so short a contact with KCy solution as is obtainable in any form of slimes plant.

The question of assaying and sampling slimes residues is one of increasing importance. With values as low as are obtained here at the present time, and which are probably lower than is attempted to be assayed in any other part of the world, it becomes a difficult question to decide as to what extent any new process is actually improving the methods in use. It is fully acknowledged that our slimes residues containing, say, ·3 dwt. of gold per ton, with half of that value in solution, will give low results unless special precautions are used. The method suggested at first sight appears to involve considerable trouble and time, but without trial its merits can hardly be discussed. I hope that the assayers will give us the benefit of their experience in this direction.

In conclusion, I may say I feel that these remarks are hardly adequate to the nature of the paper, which I hope will be widely and fairly discussed.

WESTRALIAN WET-CRUSHING PLANTS, WITH SOME NOTES ON LABOUR EFFICIENCY.

(*Read at February Meeting, 1908.*)

By Gerard W. Williams, F.C.S., A.I.M.M. (Associate).

DISCUSSION.

Mr. H. A. White (*Member of Council*): I regret that my remarks on the 3 ft. 3 in. × 13 ft. tube mill have annoyed Mr. Brett, and I hasten to disclaim any intention to give offence. I am sure the members of this Society, who are running tube mills, will agree with me that so small a machine is better adapted for experimental purposes than practical work, and that the advice to so bestow it is not ironical but sober earnest. My protest against comparing such a machine with grinding pans, under conditions (such as 75% water) quoted by Mr. Williams, will also have your support. I refrain from going into details as to cost per ton crushed to pass 150 mesh further than the bare statement that more efficient and suitable machines must necessarily reduce expenses and point out that costs of fine grinding at the Golden Horsehoe, with a larger tube mill, show a reduction on the figures quoted for the 3 ft. 3 in. mill.

With reference to the extraction at the Golden Horseshoe, I will only state that the low figures given in statement "D," though they probably mean excellent work for that particular ore, still leave a large enough margin for profitable increase, sufficient to induce the attention of the metallurgist. It may not be out of place to mention that in the past, on these fields, quite as poor figures were accepted, that expensive concentration and roasting were not unknown, and that even bromocyanide has been tried. Our metallurgists did not, however, refuse to consider the possibility of improvement, though I do not think their aim had anything to do with getting the technical world at their feet, or even snapping at their heels.

The President: Before closing the meeting I should like to say that I am glad to see that we have some members of kindred societies present, notably from the engineering societies. We are very pleased to see them here, and I hope that the proceedings have been interesting to them, and that they will favour us with their presence on future occasions.

The meeting then closed.

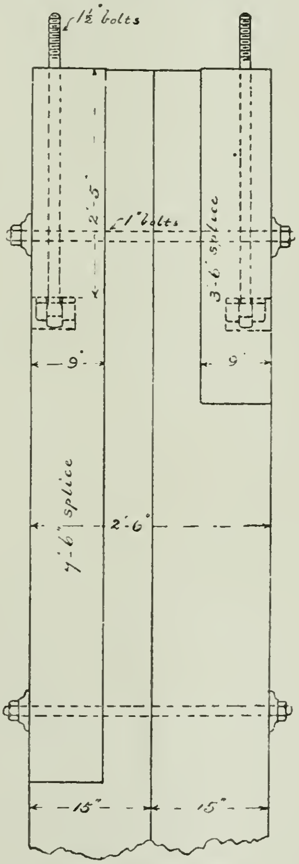
Contributions and Correspondence.

REPAIRING A MORTAR BOX PILE.

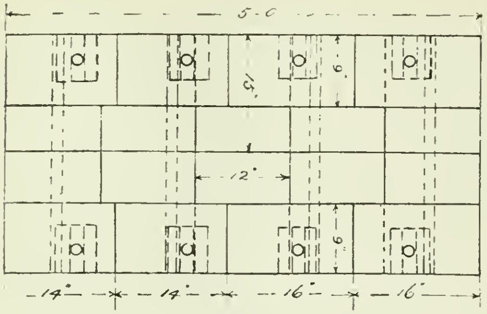
By A. Richardson (Member of Council).

Wooden mortar box piles, like other things, wear out in time and have to be either repaired or renewed; in any case, whichever plan is adopted, the making of a sound job in a short time is the object to be aimed at, so that the five stamps concerned may be out of action for the least time possible, and may subsequently continue in uninterrupted operation for the longest time possible. A satisfactory repair, answering to these requirements, having recently been effected at the Treasury mine on a pile which had been in use for over ten years, a description of it may be of interest to members.

The first symptoms of decay having reached a serious stage were given by the mortar box becoming unsteady and refusing to respond to the attentions of the holding down bolts for more than a shift or two at a time, thus indicating that the washers at the bottom of these bolts were pulling up through the wood. This state of affairs might have been allowed to continue for some little time, but there was little to be gained by so doing; there was, in the first place, a danger of cracking the box, to lessen which the feed had to be kept heavier, reducing its crushing capacity, and as the repair had to be taken in hand sooner or later it was considered advisable to undertake it at once and thus avoid all risk. The first step was to ascertain the extent of the decay and then to get everything in readiness, so that there might be no cause for delay once the five stamps were stopped. For this purpose the earth was dug away from the pile, both back and front, to expose about 8 ft. for examination; more than this is generally unnecessary, as decay rarely takes place much below the surface of the stamped in earth. The pile was then carefully examined to locate the distance of the decay downwards, while trial auger holes were put in here and there to determine its extent inwards; this having been determined, a liberal allowance of pitch pine of a suitable size, in this case 16 in. × 9 in. was then ordered and the plan of campaign drawn up. Towards the end of the night shift on Sunday the battery was dismantled in readiness for the table to be shifted out and the box to be moved back, the feeder floor having previously been shored up to permit of this latter being safely done. The two binders, back and front, above the streak sills and also the box gave some trouble to remove because of the rusted state of bolts and nuts. Two wooden straight edges of a suitable size were then nailed to the pile in a level position, both back and front, to mark the bottom joints of the splices, two similar ones being carefully nailed on the top to mark the depth of the splices inwards. Next, along each bottom straight edge, a large notch with its lower side kept carefully level was chiselled in the pile to the depth required; starting from the top, the decayed timber was then split off with big iron wedges close to the top straight edges and right down to the bottom joint; these faces were then dressed with adzes and chisels to as true a plane as time permitted, and finally rough planed, while at the same time the bottom joint was given a final dressing with a broad chisel. A pattern was then cut out of shelving, and one end dressed so as to fit into the position of the first section of the splice in one direction while the other end was made to fit in the cross direction. To this pattern was then sawn off from the log the first piece of timber required, it was then dressed again so as to fit into the next position and the next piece sawn off to it, and so on in rotation. These pieces were sawn about 3 in. longer than the finished length, and were of such a width that they broke joint with the old timber and gave a solid hold for the bolts; their bottom ends were given a run over with the plane before placing in position. All the pieces for one side were then put into position and wedged while the cross bolt holes were augered through the old holes in the solid remaining portion of the block, they were then removed and the pieces for the other side were wedged into position and their holes similarly augered; for these 1 in. bolts the holes were augered $1\frac{1}{16}$ in. All faces were then well tarred and the whole bolted up. The top of the new portion was then sawn off to straight edges to a level surface, $\frac{1}{8}$ in. above the top of the old portion, this slight overlapping is to allow for any settlement of the splices on the bottom joints and is a necessary precaution. The exact position of the box was then obtained by plumbing from the cam shaft and the holding down bolts holes were carefully marked off and augered. For these $1\frac{1}{2}$ in. bolts the holes were augered $1\frac{3}{4}$ in. The pockets for the cotters and washers were then chiselled out. Four thicknesses of tarred blanket with an extra thickness to fill the slight depression in the centre were then placed on the pile, the holding down bolts put into their places and the mortar box replaced in position and bolted down. The battery was then reassembled as rapidly as possible and the stamps dropped. All the old



END ELEVATION.



PLAN.

bolts were dressed up and used again, the two which run parallel with the long axis of the pile near the top, which came out with the portion removed, not being replaced, neither were the lower binders on the streak sills put back again. The previous sketches will help to make this description clear. A shift and a half was worked daily, this being found a better plan than running a day and a night shift; the foreman carpenter personally supervised and worked with the men the whole time, and to his close attention to detail a great deal of the success of the repair is undoubtedly due, as the whole thing depends on getting good solid joints. Three days twenty-one hours elapsed from the time the stamps were hung up till they were dropped again. This time would have been six hours shorter had not one of the king posts called for repair, and been attended to while the opportunity offered, so that the nett time of stoppage, due to this repair, was three days fifteen hours.

The cost of the work, including the foreman carpenter's time, and all labour in connection with it, including the removal of the mortar box and the replacing of it in position again, excepting that done by the shiftmen on duty, was as follows:—

Pitch pine ...	... £10	12	8			
Sundry stores ...	1	2	6			
				£11	15	2
Carpenters, 20 shifts ...	20	0	0			
Blacksmiths, 1 shift ...	1	0	0			
Millwright, 1 shift 6 hours	1	13	4			
Rigger, 1 shift ...	1	0	0			
Natives, 50 shifts ...	6	5	0			
				29	18	4
Total ...				£41	13	6

As regards the life of the repair, I can only say that a pile was repaired in this manner on the Treasury four years ago, that it has never given any trouble since, and that its splices seem as solid as on the day they were put in. Since that date nine other piles have been similarly repaired, and the work has proved perfectly satisfactory. The question will naturally arise, however, whether it might not be preferable to put in a new pile altogether instead of repairing an old one? The answer to this will depend, to a great extent, on the life of the mine, for it would not be economical to go to the extra expense of putting in work to last ten years or more, when it might be required for only half that time, or less. There might be a slight saving in time, but the cost of making the new pile, removing the old one, and placing the new one in position, would be considerably more than the cost of splicing, as the following rough estimate, worked out on a similar basis, shows:—

Pitch pine with allowance for waste 166½ c.ft. @ 4/6	£38	0	0			
Bolts, nuts and washers, 500 lb. @ 25/- ...	6	5	0			
Sundries	1	0	0			
Smith, 1 shift ...	1	0	0			
Fitter, 1 shift ...	1	0	0			
Carpenters, 32 shifts ...	32	0	0			
				£79	5	0
Millwright, 3 shifts ...	3	0	0			
Rigger, 3 shifts ...	3	0	0			
Carpenters, 8 shifts ...	8	0	0			
Natives, 54 shifts ...	6	15	0			
				20	15	0
Total ...				£100	0	0

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

A Rapid Method for the Estimation of Arsenic in Ores.—"The following method was arrived at after exhaustive trials of the usual volumetric and gravimetric processes, which were all found to be either too slow and tedious, or to require too much delicate manipulation for rapid technical work. It is suitable for sulphide or oxidised ores containing upwards of 1% of arsenic. Such ordinary constituents of the ores as lead, copper, zinc, iron, manganese, or nickel do not interfere, being either precipitated as oxides or else having no reaction with the iodine thiosulphate titration.

The reactions which take place are as follows:—

$As_2O_5 + 4KI = As_2O_3 + 2I_2 + 2H_2O$

and $2Na_2S_2O_3 + I_2 = 2NaI + Na_22S_2O_3$

The solutions required are:—

$Na_2S_2O_35H_2O$ 33·1 grm. per litre

then 1 c.c. = 0·005 grm. of arsenic.

Sodium hydrate 25% solution.

The sodium thiosulphate may be standardised either against copper or arsenious oxide.

Copper.—Take 0·3 grm. of copper, dissolve in 10 c.c. dilute nitric acid, boil off nitric fumes, add ammonium hydrate till just alkaline, boil off the excess of ammonia and then add 1 c.c. of glacial acetic acid, cool, add potassium iodide and titrate with sodium thiosulphate. Then the copper value multiplied by 75 and divided by 127·2 gives the arsenic value of the solution.

Arsenious Oxide.—Take 0·264 grm. of pure arsenious oxide, add 5 c.c. of concentrated nitric acid, evaporate to dryness and heat strongly for a few minutes, take up with 25 c.c. sodium hydrate solution, warm and then make up the bulk of the solution to 50 c.c., neutralise with concentrated hydrochloric acid and then add 25 c.c. in excess, cool, add potassium iodide and titrate until the solution becomes perfectly colourless. Test the solution with starch, and if it shows any deep blue colouration the standard must be repeated.

Treatment of Ore.—For ores containing about 20% of arsenic take 0·5 grm., place in a 12-oz. tumbler beaker and add 10 to 15 c.c. of a moderately strong solution of potassium chlorate in concentrated nitric acid, evaporate gently to complete dryness and allow to heat for a few minutes to get rid of all oxidising matter. Cool, add 10 c.c. dilute ammonium hydrate, bring to boiling to disintegrate the mass and then add 25 c.c. of the sodium hydrate solution; boil again and filter hot, and wash with hot water. The filtrate should not exceed 50 c.c., if it does, evaporate down to about 50 c.c.; neutralise with concentrated hydrochloric acid and treat as in standard.

Notes on the Method.—The proportion of hydrochloric acid to the bulk of the assay should be about one-half or three-fifths; if less acid be present a reverse reaction sets in, the arsenious oxide being oxidised to arsenic oxide, shown by the colour going and coming, during the titration. If too much hydrochloric acid be present the potassium iodide will not dissolve, and the sodium thiosulphate is decomposed and sulphur is precipitated, disguising the finish. The titration should be done slowly, especially towards the finish, and if the yellow colour returns a few drops of thiosulphate will discharge it, and the higher reading should be taken.

The starch indicator is of no use in the titration, but may be used to confirm the finish. In the strong hydrochloric acid solution the thiosulphate will not discharge the starch colour. For exact work it is advisable to take a blank assay of water and hydrochloric acid in the right proportions, add potassium iodide and titrate. This is to allow for any free chlorine being present in the acid.

If antimony be present, instead of dissolving the ore in nitric acid, it should be fused with sodium peroxide in a nickel crucible, the alkaline water extract being treated as before.

The following experiments were performed to test the accuracy of the method:—

The use of Starch gave the following Results.—50 c.c. of water + 30 c.c. hydrochloric acid + potassium iodide took 0·1 c.c. of standard thiosulphate, giving an instant finish, and, on addition of starch, no colour. Similar amounts using starch required from 0·5 c.c. to 1 c.c., and then would show no definite finish.

Effect of varying Hydrochloric Acid.—A solution of sodium arsenate in water containing the equivalent of 0·1 grm. of arsenic in 50 c.c. was used in each case.

Arsenic solution.	Hydrochloric acid.	Thiosulphate requir'd
1. 50 c.c.	10 c.c.	22·1 c.c.
2. 50 ,,	25 ,,	22·2 ,,
3. 50 ,,	30 ,,	22·2 ,,
4. 50 ,,	50 ,,	about 22·0 ,,

In (1) the colour went and came during the titration.

In (2) the finish was fairly sharp, but the starch gave a colour on standing.

In (3) the finish was sharp, and the starch gave no colour on standing.

In (4) the potassium iodide did not dissolve until well on with the titration, and towards the end the thiosulphate was decomposed, precipitating sulphur and disguising the finish.

Effect of varying Sodium Chloride:—

Arsenic sol.	Hydroch. acid.	Sod. chloride.	Titration.
1. 50 c.c.	30 c.c.	2·5 grm.	22·1 c.c.
2. 50 ,,	30 ,,	5 ,,	22·2 ,,
3. 50 ,,	30 ,,	10 ,,	22·15 ,,
4. 50 ,,	30 ,,	20 ,,	22·2 ,,

In (3) the sodium chloride only dissolved during the titration, and in (4) a considerable portion was left undissolved, rather disguising the finish.

Effect of Potassium Chlorate.—A blank assay was done using 15 c.c. of the nitric acid potassium chlorate solution and evaporating to dryness. The titration required only 0·1 c.c. of standard thiosulphate."—HARLEY E. HOOPER.—*Institution of Mining and Metallurgy, Bulletin No. 41*, Feb. 13, 1908. (A. R.)

THE THEORY AND PRACTICE OF THE IODOMETRIC DETERMINATION OF ARSENIOUS ACID.—The statements made by different authors as to the use of a neutralising agent for the arsenious acid vary considerably, the generally adopted method being to use $NaHCO_3$, and the standard test books on analytical chemistry contain incomplete and misleading statements in regard to the theory of the method and the precautions to be observed in applying it, the development of the method having been apparently purely empirical. The author therefore studied the equilibria involved, and the calculation of the proper conditions to be observed at the end point as well as the methods for securing these conditions. The reaction taking place is best expressed by the equation

$$H_3AsO_3 + I'_3 + H_2O = H_3AsO_4 + 2H^{\cdot} + 3I'$$

which is reversible completely. The constant for this as determined by Luther was computed as $3{\cdot}3 \times 10^{-2}$, and at $25° = 7 \times 10^{-2}$. Solving this further for $H^{\cdot}$ the result 1×10^{-4} was found as the upper limit of the $H^{\cdot}$ concentration. The lower limit was found to 1×10^{-9}. The best value, therefore, would evidently be the geometrical mean of these, viz., 3×10^{-7}, which differs so little from the concentration of $H^{\cdot}$ in pure water that the final conclusion was drawn, that at the completion of the titration the solution should be neutral. The phosphoric acid, carbonic and boric acid ionisation constants were considered and three combinations theoretically obtained to preserve neutrality. The final conclusions are:—

1. In titrations using $NaHCO_3$ it is absolutely essential that the solution be kept saturated with CO_2 and hence the titrations should always be made in a stoppered flask, in which at room temperature the delicacy and permanency of the end point are all that could be desired.

2. Sodium phosphate is recommended in preference to the bicarbonate, because its concentration in the solution can be controlled by the operator.

3. In titrating an unknown solution of arsenious acid the solution if alkaline is made neutral with HCl, if acid, it is neutralised with NaOH using phenolphthalein as an indicator. The neutralising agent, bicarbonate or phosphate, is then added from time to time during the titration, and when the titration is nearly complete, more of the neutralising agent is added until the total quantity added amounts to about 5 grm. of $NaHCO_3$, or 11 grm. of

$Na_2HPO_4 . 12H_2O$ for every 100 c.c. of $N/10$ iodine solution required in the titration. The volume at the end of the titration should be about 250 c.c. for every 100 c.c. of the $N/10$ iodine solution used. The Na_2HPO_4 should be added in solution from a burette.—E. W. Washburn.—*Journal of the American Chemical Society*, xxx., 1, 31, Jan., 1908. (J. A. W.)

Titration of Zinc in Alkaline Solution.—"An outline of the process which is a modification of Low's standard method follows: One-half or 1 gm. of ore is decomposed with 10 to 15 c.c. nitric acid, preferably in an Erlenmeyer flask of Low's special funnel-neck form, and as soon as red fumes cease to appear, 10 gm. potassium chlorate is added. The contents of the flask are boiled to dryness, cooled and a certain amount, 25 or 35 c.c., of the zinc wash added. The wash contains 100 c.c. ammonia and 100 gm. ammonium chloride to the litre. The flask is returned to the fire, its contents brought to a boil and poured upon the filter catching the filtrate in an 8-oz. beaker. The flask is rinsed three times with as little hot water as possible, using the mouthpiece of the wash bottle for its delivery. When this has run through, the precipitate is washed twice with hot zinc wash from a wash bottle, using enough to fill the paper to a depth of about 1 in. above the point. When this has run through a third wash of hot water, again using the wash bottle inverted, is used. After the filter has drained and the filtrate has been heated to boiling it is ready for titration.

The Titration.—The solution should be divided as usual, and after the end point is passed all of the other half is added except enough to require 1 c.c. for titration. The standard solution may then be added 1 c.c. at a time until the end point is again passed when the whole is returned to the original beaker and finished carefully 0·1 c.c. at a time. A light blue colour appears instantly with two drops excess. The indicator is prepared by adding to glacial acetic acid a few cubic centimetres of solution of ferric chloride. The colour must not be strong enough to mask the end reaction but should be sufficient to obscure any bluish tinge produced by copper in solution. Each depression of the spot plate should contain 0·3 or 0·4 c.c. of the indicator to insure acidity when 1 or 2 c.c. of the solution are withdrawn with a hollow rod. The ordinary spot plates have too shallow depressions.

Since copper is not precipitated in ammoniacal solution when potassium chlorate and zinc salts are present, its removal is unnecessary. In any event, the addition of a small amount of potassium hydrate would absolutely prevent interference."—Edgar B. van Osdel.—*Engineering and Mining Journal*, Oct. 19, 1907, p. 730. (G. H. S.)

Determination of Cupellation Temperatures.—"As a result of numerous and carefully conducted experiments with a Wilson-Mauelen pyrometer, the following conclusions were arrived at:—1st. In the case of the lead buttons, not containing any appreciable amount of copper or iron, etc., *i.e.*, pure, that a muffle temperature of at least 800° C. and, better, one of 850° is necessary to 'uncover' or start cupellation.

2nd. That this temperature may be lowered to about 770° C. during the oxidation of the greater part of the lead.

3rd. That toward the end of the cupellation, or the 'finishing' in case of silver, it must again be raised to about 830° C. in order to get a pure silver button.

4th. That the actual temperature of the cupelling lead is always appreciably higher than the muffle temperature.

5th. That the actual finishing temperature of the cupellation cannot safely be carried below about 910° C.

6th. That the greatest observed surfusion of silver was 77° C. and that this is probably very near the maximum.

7th. That silver beads finishing with surfusion are free from lead.

8th. That 'feathers' or crystals of sublimed litharge on the cupel are an indication of the proper cupellation temperature, provided the air draft is not excessive.

9th. That it is just as essential to regulate the air draft of the muffle as its temperature.

Where very accurate cupellation work is required, such as in bullion assaying and where the amount of work justifies it, a furnace designed for close temperature and air control is practically essential. In view of the recent improvement in electrically heated furnaces, in which temperatures can be rapidly and accurately controlled, and the muffle heated uniformly, practically eliminating the thermal gradient, a furnace of this type would seem best adapted for the work."—C. H. Fulton.—*Western Chemist and Metallurgist*, Feb., 1908, p. 53. (H. A. W.)

The Assay of Telluride Ores.—"The authors have made an exhaustive study of the causes of the gold losses in assaying telluride ores. Several methods for the qualitative determination of tellurium are described. Early experiments proved that the direct loss of gold by volatilisation during roasting was slight, and was not affected by the presence of mercury. The loss of gold by volatilisation during the cupellation of lead buttons, obtained from any ordinary telluride assay, was found to be extremely small, in fact not greater than in the tellurium-free lead buttons. The cupellation loss evidently depends on the ratio of lead to tellurium in the lead button; for instance, an alloy containing 60% lead, 20% gold, and 20% tellurium gave an absorption loss of 7·5% gold in cupelling. The cupellation of this same alloy with a sufficient excess of lead gave a normal volatilisation and absorption loss.

There is practically no loss of gold in the slag in scorifying highly tellurous lead, but there is a condition in scorification which greatly affects the loss in the subsequent cupellation. If a 50 gm. lead button was scorified down to two-thirds its weight, it was found that a considerable portion of the tellurium was expelled; but if the scorification was carried farther, and especially if the lead was allowed to remain under the slag for any length of time, the tellurium was reduced from the slag, re-entered the lead and enriched it to a greater extent than before scorification. The enriched button from such a scorification would undergo a heavy loss in cupellation. The authors' summary is: (1) If the lead button from the crucible assay be kept large the tellurium will be mostly eliminated in the earlier stages of cupellation and the gold loss will not be appreciably affected.

(2) A preliminary scorification of the large lead button to two-thirds of its orignal weight is advantageous.

(3) If the proportion of tellurium to gold be small in comparison with the lead in the alloy to be cupelled, the loss will be normal.

The scorification method for the original treatment of the ore is not satisfactory on account of the smallness of the charge, the danger of mechanical loss and poor decomposition of the ore. The authors recommend the crucible-fusion method, after trials of many different methods, as best suited for the assaying of telluride ores, both for gold and silver. Among the various charges recommended for different grades and kinds of telluride ores is the following for average-grade highly silicious ores: Ore, 50 gm.; red lead, 120 gm.; soda ash, 60 gm.; borax glass, 10 gm.; flour, 7 gm.

In general, it was found best to use a charge which yielded a basic slag, fusible at a low red heat, as an acid slag gave low results. The addition of fluorspar to the charge was beneficial. The lead button should be scorified large as a preliminary to cupellation as mentioned before. Salt should not be used for a cover. Fine grinding is essential in the sampling and assaying of telluride ores. The authors recommend that none of the wet methods be employed except possibly in the cases of certain rich copper or other uncommon ores. The conditions for the wet assay, if used, are described, as well as methods for working up slags and cupels."—G. T. HOLLOWAY and L. E. B. PEARSE, *Transactions, Institution of Mining and Metallurgy, Bulletin 39.*—*Engineering and Mining Journal*, March 21, 1908, p. 619. (W. A. C.)

A NEW FORM OF CHROMIUM.—"Chromium prepared in an electric furnace was found by Moissan to be slightly soluble in molten copper, and investigations of this fact led to the discovery of a new form of chromium, described in the *Revue d'Electrochimie*. This new form is crystalline, has a density of 7·1 and is chemically active; it burns with a bright flame when heated alone in the air and is attacked even by nitrogen at a red heat."—*Engineering and Mining Journal*, Feb. 15, 1908, p. 376. (G. H. S.)

STRUCTURE OF THE ATOM, *Hurter Memorial Lecture.*—"The foundation for the electronic theory of matter was stated to be the equation first worked out and published by Prof. J. J. Thomson, of Cambridge, in 1881, $M = \frac{2\,ue^2}{3a}$. This is the equation reduced to its simplest form by aid of Van der Waal's formula. By means of this equation it could be proved that the mass or inertia of an electric charge was such that, if moving with the velocity of light, its kinetic energy would exactly equal the electrostatic energy of the same charge at rest. It might be truly said then that an electric charge had the properties which we know as mass or inertia, and it thus became a possibility that no other mass, or inertia, need, or did, exist. The atom was pictured by the more advanced chemists and physicists of the present day as an assemblage of large numbers of minute electric charges called 'electrons.' These electrons were all of one sign and were negatively charged, the whole atom being maintained neutral by an equal charge of opposite sign. The negative charges or electrons were not quiescent within the atom, but were supposed to be spinning in regular orbits within the bounds of the uniform and positive charge or charges which neutralised them, and the analogy of the planets of the solar system was applicable to the internal constitution of the atom. Although the atom was so minute that a drop of sea-water contained one thousand million atoms of gold, yet it was huge compared to the 'electron,' and, if the electronic theory of matter be correct, the distances separating one electron from another within the atom, in relation to their size, must be as great as those separating the planets of the solar system. The porosity of matter, if so constituted, was then extreme, and the ultimate specks of which even the atom was composed were thus few and far between. Though one atom might contain hundreds or thousands of electrons, yet it consisted mainly of intervening space, in which there was nothing but the continuous unmodified ether. The practical applications of the 'electron' theory of matter were next illustrated, and it was shown how the arrangement of the elements in Mendeleef's periodic system, also the phenomena of mass-action and of bright-line spectra, could be explained, by aid of the newer theories. With regard to the first of these, it had been demonstrated that the similar properties possessed by the seven members of the beryllium group of elements could be attributed to the similar arrangement of their inner rings of electrons, while the atomic weights of the group were shown to be in close agreement with the total number of electrons or groups of electrons, contained within the atom. Experimental evidence had proved the truth of the theory as regarded the formation of rings within the atom. Sir Oliver Lodge closed his lecture with a reference to the vagueness and uncertainty which characterised the conception of the positive charge. The negative electron or corpuscle was now a familiar object, but the positive electron still resisted the attempts to drag him into the light of day. Progress was, however, being made, and one of the most promising openings for present-day research was this quest into the nature and properties of the positive charge."—SIR OLIVER LODGE.—*Times Engineering Supplement*, March 18, 1908. (J. A. W.)

CARBONISATION AT LOW TEMPERATURES.—"Recent researches had shown, however, that remarkable results could be achieved by carbonisation at low temperatures. It was true that by reducing the temperature from 900° C. to between 400 and 500° C. the amount of gas per ton of coal came down from 11,000 cub. ft. to about 5,000 cub. ft., but there were advantages which more than compensated for the reduction in the amount of gas. There was obtained among other things an improved yield of tar, and that tar had undergone a certain transformation. The light oils and the carbolic oils were increased five times in volume, and among the constituents was a perfect motor spirit. The creosote group of oils were also changed in character, being free from naphthalene and forming excellent enriching and fuel oils. The pitch, too, was absolutely without free carbon. Finally, the coke instead of being difficult to burn ignited more easily than coal itself, and gave a flame without the least trace of smoke. At the same time the gas produced was a good coal gas. In gas manufactured at 900° C. there was 54·5% of hydrogen, and not more than 34% of methane, while in gas manufactured at 400° C. the hydrogen had fallen to 21·2%, and the methane had increased to 60%, with a gain in the heat units of 490 to 760 to 800. It was true that only half the quantity of gas per ton of coal was obtained, but a comparison he had made between the financial results of high and low temperature carbonisation showed that the respective costs per 1,000 cub. ft. of gas delivered in the gas holder were 10·52d. for high temperature working, against 4·66d. for low temperature carbonisation, with a gain in candle power from 14 to 20, and a rise in calorific value from 592 B.T.U. to 750."—V. B. LEWES.—*Times Engineering Supplement*, March 25, 1908. (J. A. W.)

IODOMETRIC DETERMINATION OF COPPER.—"The authors used a solution of thiosulphate to estimate the free iodine. A series of tests were made in which 0·04 gm. of copper, as copper sulphate, and 0·4 gm. of potassium iodide (approximately twice the theoretical amount or an excess of 0·2 gm.) were taken: the volume of solution, at precipitation, was from 30 to 80 c.c. and at the finish from 36 to 86 c.c. These tests showed plainly that the reaction was not complete under the conditions. Another series of tests at similar dilution and with the amount of potassium iodide ranging from 4 to 60 times the theoretical amount, indicated that the reaction was practically complete. That the amount of potassium iodide used has, within limits, an influence on the result was plainly shown by further tests. As a conclusion on this point, the authors find that 2 gm. of potassium iodide is the uniform amount suitable for the precipitation of cuprous iodide in a volume of 50 c.c. of a neutral solution containing 0·2 gm. of copper; and 5 gm. as the amount if the volume is 100 c.c.

A series of blanks to determine the effect of free acid upon potassium iodide showed that a trifling amount of iodine was set free in every case, probably due to the presence of traces of iodate. Furthermore, not more than 2 c.c. of sulphuric acid or hydrochloric may safely be present with 2 gm. of potassium iodide in 50 c.c. of solution, but the presence of 1 c.c. of pure nitric acid causes error. With a dilution of 100 c.c. there may be present 3 c.c. of sulphuric hydrochloric or nitric acid (free from nitrogen oxides), with 5 gm. of potassium iodide, while 50% acetic acid may be present up to half the solution at either volume. The effect of these quantities of free acid, when copper was present in the solution, was the same as noted in the blank tests, with like volumes. It was clearly proved that a very considerable excess of potassium iodide, above the theoretical equivalent involved in the reaction is necessary, and that the necessary excess increases materially with the dilution of the solution.

The best general procedure in determining by the iodometric method for amounts of copper not exceeding about 0·3 gm. is as follows: The solution of the cupric salt, containing no more than 3 c.c. of concentrated sulphuric, hydrochloric or nitric acids (free from nitrogen oxides), or 25 c.c. of 50% acetic acid is made up to a volume of 100 c.c., 5 gm. of iodate-free potassium iodide are added, and the titration of the free iodine is made by sodium thiosulphate in the usual manner with the use of the starch indicator at the end. In case the end reaction has not appeared when 25 c.c. of the thiosulphate have been added, 2 to 3 gm. more of potassium iodide are added before continuing the titration. The error of the process, properly conducted, should not exceed a few tenths of a milligram in terms of copper."—F. A. GOOCH and F. H. HEATH, *American Journal of Science*, July, 1907.—*Engineering and Mining Journal*, March 21, 1908, p. 604. (G. H. S.)

DETERMINATION OF TIN IN TINSTONE.—Mix 1 to 1·5 gm. of the finely powdered sample with 20 times its weight of zinc-dust, and add gradually enough 16% HCl to dissolve about 90% of the zinc. When action ceases, decant off clear solution and add excess of HCl to dissolve remaining zinc and tin sponge. Filter, dilute to 100 c.c., boil and precipitate Sn by adding a rod of zinc which takes about 16 minutes. The spongy tin is detached and washed, converted to SnO_2 and weighed.—KOOMANS.—*Jaarboek van het Mijnwesen Nederlandsche Oost Indien*, vol. xxxx., p. 171. (G. H. S.)

CEMENT FROM BLAST FURNACE SLAG.—"We are informed that the Cowper-Coles process for making cement equal to Portland cement direct in one operation from blast furnace slag consists in taking the slag when still molten and raising the temperature by means of an electric resistance furnace, during which period the necessary quantity of chalk is added, and then subjecting the whole to electrolysis, which brings about certain reactions, enabling a cement to be made in practically one operation which is claimed to be equal to the best Portland cement at a very small cost compared with the present methods."—*Times Engineering Supplement*, March 18, 1908. (J. A. W.)

SEPARATION OF ALKALIES FROM MAGNESIUM.—To the faintly ammoniacal solution 40 to 80% excess $(NH_4)_2HAsO_4$ solution is added and complete precipitation effected quickly by cooling in ice and salt or adding 15% to 20% alcohol. As is removed from the filtrate by adding 10 c.c. HCl (sp. gr. 1·2) and 10 c.c. HBr (sp. gr. 1·3) or 1 to 3 gm. NH_4Br, and evaporating to dryness in a platinum dish; the residue is then ignited to remove NH_3 salts.—BROWNING and DRUSHEL.—*American Journal of Science*, vol. xxiii., p. 293. (G. H. S.)

ALCOHOL DENATURANT.—"Austria-Hungary has adopted a new denaturant for alcohol. It is a mixture of 19 parts of wood spirit, 2½ pyridine, 2½ benzol, and 1 of some other substance to be specified by the Minister of Finance. The proportion is 2½ per 100 of alcohol."—*La Nature*, April 4. — *London Mining Journal*, April 25, 1908, p. 517. (A. R.)

METALLURGY.

SPECIAL TOOL STEELS.—"In tempering modern special tool steels very high temperatures are necessary. Also great caution has to be used to prevent contact of air or carbon, as otherwise the composition of the alloy would be liable to change, with very harmful results.

The Körting electric tempering furnace is a new type especially adapted for this purpose, the feature being that the steel is heated in a bath of some fused salt, such as barium chloride, by means of a powerful electric current. An exceedingly steady temperature of 1,300° C. can be easily maintained for hours."—GORDON C. MILLS.—*Engineering and Mining Journal*, Oct. 19, 1907, p. 739. (G. H. S.)

ELECTRO-METALLURGICAL TREATMENT OF COPPER SLIMES.—This process was devised to treat copper bearing slimes from the wet concentration of sulphide copper ores, containing about 2·23% copper.

The slimes are dried, roasted at a low temperature in the neighbourhood of 440° C. for 45 minutes, in order to convert the copper to sulphate, then agitated with ¼% sulphuric acid solution, using 7 parts of solution to 1 part of ore and compressed air agitation, subsequently settling and decanting off clear solution.

Finally depositing the copper from solution electrolytically, using a current density of 1 amp. per sq. ft. to commence, and finishing at ·32 amp.

This process is estimated to produce copper at under 5d. per pound, treating only 20 tons per day and extracting 1·7% copper.

Details of the experimental work are given and are interesting in that the above temperature was determined.—D. HOLLIS, F. P. LAMMON, T. W. QUAYLE and P. D. GROMMON.—*Bulletin of the Colorado School of Mines*, Jan. 1908, p. 107. (G. H. S.)

PLATINUM AMALGAM.—"A note in the *Am. Journ. Sci.* (June, 1907) gives some interesting facts concerning platinum amalgam. It is well known that when mercury is shaken up with water the two liquids separate as soon as the agitation is stopped. Moissan has observed that it is otherwise when the mercury contains platinum in solution. After a few seconds of agitation the platinum amalgam forms a semi-solid mass of buttery consistency with a volume about five times as great as that of the original amalgam. The emulsion thus formed is so permanent that it does not appear to have changed its volume after standing at rest for a year. It resists the action of heat, for it may be heated to 100° C. without apparently changing its volume, and without any disengagement of gas. When the emulsion is subjected to a vacuum it diminishes in volume, a little water separates, and bubbles of gas are given off. The emulsion may be made by shaking 2 c.c. of distilled water to which has been added a few drops of 10 % solution of platinic chloride."—*Engineering and Mining Journal*, Oct. 5, 1907, p. 630. (G. H. S.)

METALLURGY OF THE KALGOORLIE GOLDFIELD.—Members wishing to become acquainted with the present day metallurgy of the Kalgoorlie goldfield should consult this article, which is too full of detail to be abstracted.—G. W. WILLIAMS.—*Engineering and Mining Journal*, Feb. 15, 1908, pp. 345—350. (G. H. S.)

CYANIDE PRECIPITATION AND CLEAN-UP AT THE PORTLAND, COLORADO, MILL.—"The precipitation room is large and well lighted, and contains eighteen rows of zinc boxes, each row having six compartments 4 ft. by 4 ft. by 4 ft. and pyramidal in shape at the bottom. A 14 mesh wire screen supported by 2-in. angle iron riveted on the inside 20 in. above the lowest point of the pyramid, forms the false bottom. The solution enters the first box through a false compartment, flows under the screen, then up through the zinc and over into next compartment; the distance from the screen to the overflow of each compartment is 28 in. These boxes are supported on timbers high enough above the floor below (which is of cement) to permit a man to stand up to work about them.

Beneath each row of boxes there is a 12 in. by 12 in. wooden launder, which carries the discharge to a launder of the same dimensions, which discharges into a steel tank with capacity for holding all of the solution from one row of zinc boxes during a clean-up. This tank is connected to a 2-in. centrifugal pump, which lifts the contents 30 ft. to a lead-lined tank (8 ft. in diameter and 10 ft. high). Over this tank there is built a hood having a 12 in. by 12 in. opening in the centre for the escaping fumes during a clean-up.

These gases are assisted out of the hood by means of an air jet. Air is also used to agitate the mixture in the tank when cutting down the zinc.

Of the eighteen sets of zinc boxes, we are using twelve, each compartment of which is packed with 200 lb. of zinc shavings previously dipped in a 2% acetate of lead solution. The total amount of zinc used to pack the boxes is 14,400 lb. Strong solution flows through six rows of boxes—the strength of cyanide per ton of solution being ·6 lb., and the protective alkalinity ·5 to 1·0 lb. Weak solution flows through the other six boxes—the strength of cyanide per ton of solution being ·3 to ·4 lb., and the protective alkalinity, neutral to ·2 lb.

On the strong side, from 500 tons to 600 tons of solution flows through in twenty-four hours, while on the weak side from 800 tons to 1,000 tons—a total of from 1,300 tons to 1,600 tons daily. The gold contents in the solution entering the strong side varies from 20 cts. to 40 cts. per ton, while the weak solution contains from 10 cts. to 20 cts. Our precipitation is remarkably good—showing only a trace in gold and 4 cts. being high.

A white precipitate forms on all the boxes at the head compartment, this condition being more pronounced on the weak side—in fact, so great is the amount formed on this side that it necessitates a separation of this muck from the fine zinc when we clean up. It readily falls off when the zinc is disturbed. It consists of calcium ferro-cyanide, zinc hydrate, calcium sulphate, calcium carbonate, and fine slime from the sand filters.

The formation of this white precipitate in such large quantities on the weak side is a rather curious fact, when it is remembered that the strong side carries the strongest percentage in lime. The only explanation I can give for it is that the solution is so weak in both cyanide and protective alkalies that it has not sufficient strength to hold it in solution when in contact with the zinc.

Our zinc is received in car lots from the Matthiessen & Hegeler Zinc Company, La Salle, Ills., and is called the No. 9. It comes in rolls of 600 lb., the sheets being 3 ft. wide and 7 ft. long. Twenty of the sheets are rolled up on a 6-in. mandrel. The lathe is an ordinary one, not especially built for zinc cutting. One man cuts from 350 lb. to 450 lb. shavings in eight hours, and the cost of the cut zinc per lb. is 10½ cts.

When clean-up time arrives we start on the weak side, take out all the zinc in the second compartment, then a tight fitting 4 mesh screen is placed on the box. The zinc is removed, is put over the screen, while that portion remaining on top is placed in tubs until the compartments are cleaned, and is then put back in boxes. The first compartment is in the meantime being drained of solution from the bottom and all of the zinc is passed over the screen in the same manner. What fine zinc remains on top of the screen in the bottom of the compartment after being screened through the top one is shovelled up and held in tubs until the weak boxes are cleaned. After the fine zinc is taken off the screen in the bottom of the box the screen is raised and all dirt and fine zinc is washed out of the bottom into the afore-mentioned launder. The main launder, where it discharges into the tank (earlier mentioned) has a board nailed part way up on the end, so that when the boxes are drained, the heavy particles stay in the bottom of the launder, while the light particles overflow into the tank. This, no doubt, may appear to be a crude way to make a separation of this product, but it gives the results. This light muck and cyanide solution is pumped to the lead-lined tank above, where it is left to settle, and the cyanide solution is decanted off by means of a decanting line into the head of one zinc box, where the solution goes back into the system. The white muck left behind in the tank is then let out at the bottom and washed into a filter-press, which, when full, is blown with air, cleaned out, dried, sacked, and shipped to the smelter. Its value averages from 15 oz. to 30 oz. gold, and the same in silver per ton. The fine zinc caught in the launder referred to, and in the zinc box, is cut down in the usual way with sulphuric acid, washed, and sent up to the filter-press, from which it is later removed and sent to the melter. All of the acid solution from this treatment goes to a tank located outside the mill; zinc dust is added, and the mixture agitated with air and left to settle until the next clean-up, when the solution is decanted off. The tank is cleaned up every three or four months, and the amount of gold caught here, which otherwise would have gone down the hill, is astonishing. It more than pays for the expense of cleaning up.

To show just what this separation of muck, or white precipitate and fines or slimes, means, I will give some figures. The first few times we cleaned up everything was acid treated and sent to the melter. The total dry weight was 1,200 lb., which took several days to melt, and yielded a very small button as a result. By making this separation the melter gets from 100 lb. to 150 lb., which when melted gives very respectable buttons, the remaining 1,000 or 1,100 lb. not being worth melting, and being a good product for the smelter. We have no separation of this kind to make on the strong side; everything goes into the acid treatment tanks, is then filter-pressed and sent to the melter. The value of the slimes per pound after roasting is from 1 oz. to 3 oz. gold, and the same in weight of silver: in other words, the gold and silver are of about equal weights, the former being perhaps slightly in excess." —J. M. Tippett.—*London Mining Journal*, Feb. 22, 1908, p. 215. (A. R.)

Oil Engines for Small Batteries.—"To obtain cheap crushing cheap power must be available. Power costs mount up in various ways, such as labour, fuel, stores, repairs, etc. The two chief items in any plant are labour and fuel. In a small steam plant these two items will often be about equal. It is here that the oil engine shows the advantage, the labour required being very small, and the fuel bill low. Let us take, as an example, a five-head battery, with stamps of about 1,000 lb weight. This is a very useful type of plant for a small mine or a new district. It should crush 75 tons of average quartz in a week through a screen of about 1,000 mesh (woven wire).

Now, compare a week's expenditure running a steam driven plant against one driven by an oil engine. Take the steam driven plant: The little battery will require an engine capable of developing about 12 h.p., and a boiler rated a little higher, to allow steam for a return water pump. This boiler will consume in the week of 144 hours about 12 cords of wood, which will cost £1 per cord, or £12 per week for fuel. It will also require the attention of three certificated engine-drivers, each at £4 per week, and the battery will require three attendants, each at £4 per week, or a total of £36 per week for fuel and labour, or, on an output of 75 tons per week, fuel and labour costs amount to 9s. 7d. per ton of ore crushed.

Now, take the oil engine: This will also require to be about 12 h.p., and will require about 100 gallons of kerosene per week, costing about 1s. 2d. per gallon, or, say, £6 per week for fuel. The three certificated engine drivers will not be required; the three battery attendants at £4 per week will also see to the oil engine; thus the total wages and fuel amount to £18 per week, or 4s. 9d. per ton on 75 tons per week. We have thus with the oil engine a saving of £18 per week in fuel and wages, or, say, £900 on a full year's run.

Sundries, such as mercury, screens, etc., will be the same in any case, and will have to be added to the above, say, 6d. per ton treated. In the matter of upkeep, the balance is all in favour of the oil engine, there being no boiler maintenance charges, nor lying off and preparing for inspection expenses, fees, etc. In districts where boiler feed water has to be condensed, there has to be added to power costs the fuel used in condensers, maintenance, and renewals of condenser boilers and coolers. These charges may add 50% to power costs with a steam plant, in which case the saving noted above in favour of the oil engine would be still greater.

Granted that the battery has a good water supply, 1½ dwt. per ton should cover crushing cost at a five-stamp battery driven by an oil engine.

In laying out a small plant as above, where it is intended that it is to be run by one man per shift, the writer would always recommend placing the oil engine beside the table. By this means the plant is made compact, and the man in charge has everything under his eye. A five-stamp battery intended to be driven by an oil engine should always be provided with a counter shaft and belt tightener."—W. F. Smith.—*Australian Mining Standard*, March 11, 1908. (W. A. C.)

Dwight and Lloyd's Sintering Process.—This is the latest development of the modern blast-roasting process by which sulphide ores in a fine state of division are rendered fit for blast furnace treatment by blowing air through them, and so developing sufficient heat to cause the material to sinter together.

The apparatus consists of a drum formed of a pair of circular rims of iron carrying a set of cast iron herringbone grates between them and parallel to the axis; this rests on rollers like some tube mills and is caused to turn slowly by the friction of the drive rollers. Inside the drum and occupying the top quadrant of the circle is a stationary suction box connected with a suction fan, giving a suction of 4 oz. The material to be sintered is fed in a 4-in. layer upon the grated face of the drum from an overhead ore hopper, and immediately passes under an igniter, which may be a series of gas jets, etc. The roasting action so begun is maintained by the air sucked through the moving layer of ore as it passes across the suction box, which takes about 20 minutes. Further on the layer of sinter is stripped off automatically by a comb-shaped grizzly. The capacity is about 1 to 1¼ tons per day per sq. ft. of effective grate area; power required was 12 h.p. for an output of 30 tons per day, first cost was much less than reverberatory roasting furnaces, and scarcely any loss of lead occurred by volatilisation.—A. S. Dwight.—*Engineering and Mining Journal*, March 28, 1908, p. 649. (G. H. S.)

Passivity of Gold.—"In potassium cyanide solution, gold dissolves at the anode as a univalent ion. Passivity develops in commercial potassium cyanide solution at a small concentration, a brown colour, due to paracyanogen, being noticed in the solution, but in pure potassium cyanide a much higher concentration is necessary. The presence of the sodium salt in the commercial substance is responsible for this difference and the cause of passivity appears to be the formation of an invisible protective coating of sodium aurocyanide on the anode, this salt being much less soluble than that of potassium (a fact which explains the superior efficiency of potassium cyanide over sodium cyanide for the purpose of gold extraction). A similar result is noticed in the case of nickel, which dissolves quantitatively in a solution of potassium cyanide but is rendered passive, it is now found, in the presence of sodium cyanide. The authors would extend this explanation of passivity to all known instances of the phenomenon. When, in the neighbourhood of the anode, the solubility product of the ions of the protecting substance is reached (hastened in this case by the presence of more sodium ions), the protective coating is formed, rendering the anode passive."—A. Coehn and C. L. Jacobson, *Zeitschr. Anorg. Chemie.*—*Mining and Scientific Press*, Mar. 28, 1908, p. 431. (W. A. C.)

Volatilisation of Silver and Gold.—"It is known that both of these metals can be vapourised; it can easily be done in the electric arc. If they are placed in a vacuum, in quartz vessels, they show signs of metal vapourising at 680° and 1,070°, respectively, although what vapour tension this corresponds to we do not exactly know, but we can surmise it to be a small fraction of a millimetre. If heated still higher in a vacuum, they show the phenomenon of ebullition, or boil, at 1,360° and 1,800°, respectively, although here again we do not know what pressure this corresponds to, except it is the pressure of the melted metals above the spot where the vapour commences to form beneath the surface, which might be 5 to 10 mm. of melted metal, say, 10 mm. of mercury, if the bath of metal were shallow. From these data, obtained by Schuller, Krafft and Bergfeld, it is estimated tht the boiling points of the two metals under atmospheric pressure are 2,040° and 2,530°, respectively, on the assumption that the temperature interval between first signs of vapourisation in a vacuum and boiling in a vacuum is equal to the interval between the latter temperature and the ordinary boiling point. (This is said to be true of mercury.)

O. P. Watts has estimated the boiling point of silver and gold as 1,850° and 2,800°, respectively, meaning probably at atmospheric pressure; but his estimate is based on such assumptions that they cannot lay claim to as much accuracy as the data and estimates above cited.

If we go back to the very probable rule, that at equal fractions of the normal boiling points, expressed in absolute temperatures, metals have the same vapour tensions, we can compare silver and gold with mercury, whose vapour tension is known through a wide range, and get quite probable values for the vapour tensions of these metals through a large range of temperature. The following table is from nearly zero tension to about the temperature at which ebullition is noticeable in vacuum.

Tension of Vapour mm. of Hg.	Mercury C°	Lead. C°	Silver. C°	Gold. C°
0·0002	0	625	729	942
0·0005	10	658	766	987
0·0013	20	691	802	1,031
0·0029	30	724	839	1,075
0·0063	40	757	876	1,120
0·013	50	790	913	1,165
0·026	60	822	949	1,209
0·050	70	855	986	1,254
0·093	80	888	1,023	1,298
0·165	90	921	1,059	1,343
0·285	100	954	1,096	1,387
0·478	110	987	1,133	1,432
0·779	120	1,020	1,169	1,476
1·24	130	1,053	1,206	1,520
1·93	140	1,086	1,243	1,565
2·93	150	1,119	1,280	1,611
4·38	160	1,151	1,316	1,654
6·41	170	1,184	1,353	1,699
9·23	180	1,217	1,390	1,743

Inspecting the above table, we see that apparently about 0·0002 mm. of mercury tension is sufficient to make a metal show signs of vapourisation. The corresponding temperatures at which silver and gold show 0·0002 mm. tension are, from the above table, 729° and 942°, respectively, whereas it is said to have been observed for these metals at 680° and 1,070°. The divergence is not wide, considering the lack of exact data involved.

Mercury is said to show ebullition in a vacuum at 180°, lead at 1,250°, silver at 1,360°, and gold at 1,800°. From the above table we see these metals having the same vapour tension at 180°, 1,217°, 1,390° and 1,743°, respectively. The agreement is encouraging.

The table in continuation is for temperatures from those causing ebullition in a vacuum to those required to boil the metals at atmospheric pressure:

9·23	180	1,217	1,390	1,743
14·84	190	1,250	1,427	1,788
19·90	200	1,283	1,463	1,832
26 35	210	1,316	1,500	1,877
34·70	220	1,349	1,537	1,921
45·35	230	1,382	1,574	1,965
58·82	240	1,415	1,610	2,010
75·75	250	1,448	1,647	2,055
96·73	260	1,480	1,684	2,099
123·	270	1,513	1,720	2,144
155·	280	1,546	1,757	2,188
195·	290	1,579	1,794	2,233
242·	300	1,612	1,830	2,277
300·	310	1,645	1,867	2,322
369·	320	1,678	1,904	2,366
451·	330	1,711	1,941	2,410
548·	340	1,744	1,977	2,455
663·	350	1,777	2,014	2,500
760·	357	1,800	2,040	2,530

For tensions of metallic vapours over 1 atmosphere, such as may easily occur in high temperature work, particularly in electric furnaces, the following data may be useful:

1·0	357	1,800	2,040	2,530
2·1	400	1,951	2,197	2,722
4·25	450	2,116	2,380	2,945
8·	500	2,280	2,564	3,167
13·8	550	2,445	2,747	3,390
22·3	600	2,609	2,931	3,612
34·	650	2,774	3,114	3,835
50·	700	2,938	3,298	4,057
72·	750	3,103	3,488	4,280
102·	800	3,267	3,665	4,502
137·5	850	3,436	3,841	3,725
162·	880	3,525	3,958	4,858

The last table may not be of much immediate practical value, but it is interesting scientific information. The preceding tables are, however, technically and commercially important. They point particularly to the wastefulness of melting silver or gold in open furnaces where furnace gases pass over the metal. Such gases, at temperatures above 700° for silver and 950° for gold, *i.e.*, even passing over unmelted metal, can cause volatilisation and loss of weight, because they evaporate the metal on exactly the same principle that a current of dry air evaporates ice or water. They also explain why silver volatilises with lead in the cupellation operation. At 1,000° C. lead has a vapour tension of about 0·62 mm. of mercury, and silver about 0·07 mm., and, therefore, a considerable loss of silver is sure to occur with the lead vapours passing off. Gold at the same temperature has only 0·0007 mm. tension, and, therefore, proportionately less of it is lost, say, only one-fiftieth as much as the silver loss, reckoning from the proportionate tensions and the relative densities of the two vapours.

In calculating such metallic vapour losses it is important to remember that the metals are monatomic when in the state of vapour, and that the molecular weights of their vapours are simply their atomic weights. The hypothetical weight of a cubic metre of such metallic vapour at standard conditions, 0° C. and 760 mm. pressure, is therefore,

$$0{\cdot}09 \text{ kg.} \times \frac{\text{atomic weight}}{2},$$

and from this theoretical datum the weight of any volume at any temperature and any tension can be calculated.

Illustration.—Calculate the weight of silver vapour contained in 1 cub. metre of furnace gases, if saturated with silver vapour and at 1,100° C.

Solution.—Since the tension of silver at this temperature is 0·28 mm., the question resolves itself into this: What is the weight of 1 cub. metre of silver vapour at 1,100° and 0·28 mm., pressure? One cubic metre at standard conditions would be $0{\cdot}09 \times (108 \div 2) = 4{\cdot}86$ kgm.; therefore, at these given conditions, it contains:

$$4{\cdot}86 \times \frac{273}{1{,}100 + 273} \times \frac{0{\cdot}28}{760} = 0{\cdot}0035 \text{ kgm.}$$
$$= 3{\cdot}5 \text{ gm}$$

It is true that the silver vapour, being heavy, mixes slowly with the furnace gases, but, nevertheless, the gases rushing over and coming in contact with the silver may become nearly saturated with this vapour and carry considerable away. In the Bessemerising of copper matte to blister copper, by blowing air directly through it, as much as 30% of all the silver present may be thus vapourised from the bath as soon as the silver has taken the metallic form.

Every smelter and refiner of the precious metals should be familiar with these facts, and draw conclusions from them useful to the conduct of his business.

The specific heat of these metallic vapours is 0·225 per cub. metre (calculated to standard conditions), and the latent heat of vapourisation is approximately twenty-three times the normal boiling point expressed in absolute temperature for one atomic weight of the metal—as explained in Part I. of these calculations:

For silver and gold we would have the latent heats

Ag, 23 (2,040 + 273) = 53,200 Cal. per 108 kgm.
= 493 ,, 1 kgm.
Au, 23 (2,530 + 273) = 64,470 ,, 197 kgm.
= 327 ,, 1 kgm.

The above latent heats of vaporisation are for boiling at normal atmospheric pressure; for vaporisation at other pressures and corresponding temperatures, correction must be made for the difference in specific heats of the metallic vapour and liquid metal."—Prof. Richards.—*Electro-Chemical and Metallurgical Industry*, March, 1908, pp. 115-6. (A. McA. J.)

Calorific Deterioration in Stored Coal.—"The University of Illinois has issued a report of some experiments on the loss of calorific value of coal during storage. The coal was stored under four sets of conditions—(1) out door conditions, (2) dry atmosphere at a temperature of from 85 to 125° F., (3) same as (2) but drenched with water two or three times a week, and (4) submerged in ordinary water at about 70° F. The tests show that the submerged coal is practically free from deterioration, but that the other methods of storage all involve loss of heating value of from 2 to 10%, and also cause actual weathering of the coal. In this connection it was found that even in a dry atmosphere this weathering still took place, but practically ceased in all cases after about five months. The loss in heating value went on for about seven to nine months."—*London Mining Journal*, Mar. 20, 1908, p 558. (A. R.)

Unsuspected Minerals in Gold Ores.—"The presence of unsuspected minerals in gold ores may be learned by the analysis of the mattes resulting from the refining of the precipitate from the cyanide solutions. According to W. P. Headden (*Proc.* Colo. Sci. Soc., Vol. VIII., pp. 349—362), there is a trace of cadmium in some of the Cripple Creek ores heretofore unsuspected and in the Camp Bird ore (Ouray, Colo.) there is more tellurium than formerly was thought, and also some selenium hitherto unsuspected."—*Engineering and Mining Journal*, Mar. 21, 1908, p. 604. (W. A. C.)

MINING.

Quantitative Estimation of Dust in Air.—"In spite of the injurious action on the health of the presence in the atmosphere of dust and soot, there is, according to Prof. Martin Hahn, no satisfactory method at present available for determining the amount of dust in any sample of air. He has prepared therefore a small and portable apparatus for the practical testing of air to ascertain the amount of dust and soot it may contain. The apparatus is described and illustrated in the *Gesundheits-Ingenieur* for March 14. This consists essentially of a double cylinder pump with a counting arrangement which is worked by means of an electro-motor driven by accumulators. The counter indicates the volume of air passing through the apparatus by means of the aspirator. The air is made to travel through a filter tube, which is partly closed by collodion-wool, and, in order to estimate the quantity of dust in a given volume of air, it is proposed to dissolve the wool in a mixture of two parts of ether and one part of alcohol and then to compare the turbid fluid containing the soot and dust in solution and suspension with a standardised solution prepared from previous tests with known quantities of dust. Other methods of using the apparatus are suggested. The apparatus complete weighs about 15 lb. with the four accumulators."—*The Times* (London) *Engineering Supplement*, March 25, 1906, p. 6. (W. A. C.)

The Selection of Carbon for Diamond Drill Bits.—"Great care and experience are necessary in selecting carbon for diamond drill bits. The stones should be as bulky as possible and nearly round, as thin edges and irregularities make firm setting difficult and the projections wear away quickly. A small blocky stone is more economical than a larger irregular stone. Practical drillmen often prefer whole or natural stones to fragments of a larger 'broken' diamond, as they are water worn and somewhat rounded. But buyers ordinarily prefer the broken stones, as the fresh break enables them to determine accurately the quality of the stone and its fitness for drilling purposes, while, in the majority of cases it would be impossible for them to judge the hardness and grain of the weather-stained natural stone.

Broken stones with a closely knit structure and an appearance like fine steel when broken, and grey or greenish grey in colour, are hard and tough, and each grain will acquire a polish under wear. Stones with a coarse or porous grain like coke and which are dull and lustreless in colour will wear rapidly. The ideal shape for a new broken stone is the cube.

In selecting whole carbons or natural stones the preference should be given to those which have a bright or polished grain; those which are coarse in texture or show wear, but no polish, should be avoided. Good carbon will stand heavy pressure but will break under a blow, hence care should be taken by the operator when running through strata of varying hardness. Crystalline or semi-crystallised stones should be avoided, as they are apt to crush under pressure. Bortz or imperfect white diamonds are little used, in spite of the fact that they cost only about one-fourth as much as carbon, as they contain more or less flaws which weaken the stone and thereby restrict its use to coal prospecting and some quarry work in the Central States. Experience has shown that the person unfamiliar with carbon and diamond drilling should rely on the experience of the drill manufacturer to supply proper stones, as he has everything to gain by equipping his drill with good carbon."—*Engineering and Mining Journal*, Oct. 12, 1907, p. 695. (G. H. S.)

MERITS AND DEMERITS OF AIR-HAMMER DRILLS.—"With the later types of these machines the average depth of hole drilled in the Cripple Creek district is greater than the average for a 2¼ in. piston drill. The depth of hole is limited in practice only by the length of steel that it is convenient to employ. The main drawback seems to be that air-hammer drills are not readily adapted to flat or down holes. For this kind of work it is necessary to use hollow steel with either air or water, or both, passing through it to clear the hole. If holes are drilled dry the amount of dust raised is sufficient to condemn the practice at once, while, if a water attachment is used, it is necessary to have water under pressure brought to the machine. For this purpose either a water pipe must be put in or a water tank used, the pressure being obtained through connection with the air pipe.

In driving drifts it is generally necessary to make a set-up; and thus is lost one of the great advantages obtained in overhead stoping. For these reasons the machine is not likely to replace piston machines for drifting or breasting. If they do replace piston machines for this kind of work they must show a considerably greater drilling efficiency in order to offset the disadvantages.

I have not had enough experience in this kind of work to speak with any authority upon this phase of the subject. For overhead work, however, I do not hesitate to say that a properly constructed air-hammer machine, with the hammer of the same size as the piston of the piston machine with which it is compared, will drill approximately double the footage per shift. Of course, each machine is to use the same size bits. The character of the rock drilled does not affect this comparison except that wet sticky ground will be somewhat more favourable to the piston machine. In the Cripple Creek district the companies using air-hammer drills claim that in overhead drilling a saving of approximately 40% is made by using air-hammers instead of piston drills."—G. E. WOLCOTT.—*Engineering and Mining Journal*, Feb. 15, 1908, p. 351. (G. H. S.)

RESCUE APPARATUS.—A list of papers, etc., which have been written on the subject of rescue appliances:

The Walcher Pneumatophore and the Employment of Oxygen for Life Saving Purposes, Feb. 2, 1898.

A Joint Colliery Rescue Station, March 30, 1901.

The Work of Joint Colliery Rescue Station, Oct. 8, 1904.

Rescue Apparatus and the Experience gained at the Courrières Collieries by the German Rescue Party (Shamrock), June 14, 1906.

A New Apparatus (Weg) for Rescue Work in Mines, June 14, 1906.

The Pneumatogen, the Self-Generating Rescue Apparatus, July 19, 1906.

Liquid Air and Its Use in Rescue Apparatus (Aerolith), Dec. 8, 1906.

Experimental Gallery at Altofts Collieries, March 23, 1907.

The Use and Care of Oxygen Breathing Apparatus, April 9, 1907.

Demonstration of Rescue Apparatus, Felling, Aug. 31, 1907, April 11, 1908.

Notes on Recent Experience in the Practical Use of Rescue Apparatus (Draeger), Feb. 4, 1908.

Breathing Apparatus (Fleuss) for Use in Mines, Feb. 5, 1908.

(The foregoing papers are printed in the *Trans. Inst. of Mining Engineers* on the dates given.

Rescue Work in Collieries, *Colliery Guardian*, April 10, 1908.

First Report of the Royal Commission on Mines, 1907.

(T. J.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

ORGANIC CHEMISTRY. By J. B. COHEN, Ph.D., B.Sc. Pp. viii. + 632. Price 21s. (London: Edward Arnold, 1907.)

"This book will be welcomed by all teachers who have classes of advanced students in organic chemistry, or who themselves require an advanced reference book. While on the Continent there are many books on organic chemistry, there is rather a dearth of such books in this country. It is true there are several more or less elementary text-books, which are excellent in themselves, but there are very few which enter more deeply into the subject. Dr. Cohen has now supplied that want, and given us a very useful work. Owing to the excellent way in which the references are supplied the book will become to a considerable extent one of reference.

It really consists of a series of essays upon different branches of organic chemistry, each subject being treated historically and brought right up to date, so that the gradual evolution of the theory and practice is elaborated. Being written in essay form, there are, of course, a good many subjects left out which some might desire to have included; but taken altogether, the book contains a very large amount of useful matter. Stereochemistry is very much to the fore, there being sections on isomerism and stereochemistry, stereochemistry of unsaturated and cyclic compounds, stereochemistry of nitrogen, isomeric changes, steric hindrance, etc. The sections upon fermentation and enzymic action and upon the proteins deserve mention, and that upon the terpenes and camphors is also of particular interest.

The book is well got up and printed, but the binding might very well have been rather stronger. This work is quite sure of a hearty reception, and will certainly lighten the labours of lecturers in the higher branches of organic chemistry."—*Transactions of the Faraday Society*, March, 1908, p. 223. (H. A. W.)

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 244/08. James Spry. Improvements in means for changing vehicles (such as from skip to cage or *vice versa*) employed in mine shafts and the like. 23.6.08.

(P.) 245/08. John Ninnis Michell. Improvements in safety brake arrangements for mine cages, skips, lifts and the like. 23.6.08.

(C.) 246/08. Frederick Gross. Improvements applicable to locomotives or other steam engines. 23.6.08.

(P.) 247/08. John Davies. Improvements in furnaces for smelting metals. 24.6.08.

(C.) 248/08. Otto Lietzemayer. Improvements in internal combustion motors. 26.6.08.

(C.) 250/08. John Russell (1), Joseph Marsicano (2), Joseph Amorosa (3). Improvements in electrical switches for use with signal recording apparatus for mines and the like. 29.6.08.

(P.) 251/08. Frederick William Denhigh (1), Job Smith (2). Improvements in means for recovering gold from alluvial material tailings and the like. 29.6.08.

(P.) 253/08. Alexander John Arbuckle (1), Alfred Osborne (2). Improvements in or appertaining to means for separating crushed ore products from liquids. 1.7.08

(P.) 254/08. Frederick Adamson. Improved diamond washing apparatus. 1.7.08.

(C.) 255/08. Kynoch, Ltd. (1), J. P. Udal (2). Improvements in wrappers for blasting explosives. 1.7.08.

(P.) 256/08. A. C. Burmester (1), N. W. Smith (2). Automatic centrifugal spray for liquids or gases. 2.7.08.

(C.) 257/08. Robert Hallowell Richards. Improvements in apparatus for classifying minerals. 2.7.08.

(C.) 258/08. Robert Hallowell Richards. Improvements in jigs. 2.7.08.

(P.) 259/08. Gustaaf Adolf van der Eertwegh. A new motor power. 2.7.08.

(P.) 260/08. Charles Henry Perrins. A substitute for carbon or diamonds used in drilling or for similar purposes. 2.7.08.

(P.) 261/08. Wilhelm Mauss. Improvements in direct acting engines. 3.7.08.

(P.) 262/08. Benjamin Waites. Hydraulic classifier and concentrator. 3.7.08.

(P.) 263/08. Karl Ruschenbaum. Improvements in rock drilling machines. 4.7.08.

(P.) 264/08. Ernest Slatter Day. Improvements in slime treatment plant. 4.7.08.

(P.) 266/08. Diederick Cornelius Johannes Enzlin. Improvements in amalgamators. 6.7.08.

(P.) 267/08. Alexander Gemmell. Acetylene gas generator. 6.7.08.

(P.) 268/08. Charles William Dowsett. An improved rapid method for the detection of small quantities of gold in the cyanide solutions used to extract gold from its ores. 6.7.08.

(C.) 269/08. John Edward Wood. An improved feed water heater and purifier for steam generators. 7.7.08.

(C.) 270/08. Charles Shields McIntire. Improved construction of apparatus for casting links in chain form. 7.7.08.

(P.) 271/08. Frederick Victor William Swanton (1), Robert Conolly (2), Robert Lygertwood Esson (3). Improved detachable single or multiple cam for round, square or eliptical shafts in machinery. 9.7.08.

(P.) 273/08. George Robert Simms. A gravitation conveyor. 10.7.08.

(C.) 274/08. Matthew Yarrow. Improvements in machines for the manufacture of earthenware pipes. 10.7.08.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

Arkell, D. J., *l/o* Rhodesia; Niekerk Syndicate, Ltd., P. O. Box 39, Klerksdorp.
Blaine, G. R., *l/o* Cleveland: Kokstadt, C.C.
Cameron, S. J., *to* P. O. Box 2772, Johannesburg.
Cameron, W. McC., *l/o* Germiston; P. O. Box 1156, Johannesburg.
Cannon, A. B., *l/o* Krugersdorp: Kidson Reef Syndicate, Eureka City, Transvaal.
Cogle, C. T., *to* 36, Crown Road, Fordsburg.
Edwards, A. L., *l/o* Jeppestown; P. O. Box 727, Johannesburg.
Evans, Lewis, *l/o* Benoni; Robinson Central Deep, Ltd., P. O. Box 1145, Johannesburg.
Galbreath, N. M., *to* 20, Union Street, Greenock, Scotland.
Giblin, N. E., *l/o* Kalgoorlie; c/o L. K. Ward, Esq., Geological Survey, Launceston, Tasmania.
Harris, J. H., *to* P. O. Box 30, Cleveland.
Heeley, A. W., *l/o* West Nicholson: Crocodile Reef, Essexvale.
Krige, W. A., *l/o* Rietfontein; Consolidated Langlaagte Mines, Ltd., P. O. Box 15, Langlaagte.
Lawrie, E., *l/o* Germiston; Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg.
Milligan, J. T., *l/o* Bertrams; Niekerk Syndicate, Ltd., Klerksdorp.
Mills, L. D., *l/o* Deadwood, South Dakota; 502, Rincon Building, 143, Second Street, San Francisco, U.S.A.
Murphy, T. D., *to* Promontorio, Durango, Mexico.
Parker, W. F., *to* Village Deep, Ltd., P. O. Box 1145, Johannesburg.
Pead, C. H., *l/o* East Rand; P. O. Box 252, Bulawayo.
Smith, M., *l/o* Germiston; Worcester Exploration and G. M. Co., Ltd., P. O. Box 85, Barberton.
Spengel, H., *to* Victoria Falls Power Co., Ltd., P. O. Box 3498, Johannesburg.
Wilkinson, W. Fischer, *to* 43, Threadneedle Street, London, E.C.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. IX. AUGUST, 1908. NO. 2.

Proceedings
AT
Ordinary General Meeting, August 15, 1908.

The Ordinary Monthly General Meeting of the Society was held in the Chamber of Mines, on Saturday, August 15th, Mr. R. G. Bevington (President), in the chair. There were also present :—

43 Members : Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, K. L. Graham, A. Richardson, G. O. Smart, Prof. G. H. Stanley, H. A. White, Prof. J. A. Wilkinson, W. A. Caldecott, Prof. J. Yates, W. Beaver, H. D. Bell, P. Carter, F. W. Cindel, A. A. Coaton, M. H. Coombe, W. M. Coulter, M. Cullen, L. Evans, J. Gray, J. H. Harris, S. A. Herbert, A. Heymann, J. A. Jones, J. Kennedy, Hy. Lea, G. Melvill, J. E. Metcalf, E. Pam, J. F. Pyles, W. H. Roe, N. Rogers, O. D. Ross, C. E. Rusden, H. Scarf, W. S. Sharwood, R. Stokes, H. Taylor, A. Thomas, A. D. Viney and F. W. Watson.

6 Associates : Messrs. A. Avent, B. Ellis, C. B. Hilliard, H. A. Owen, W. A. C. Tayler and W. Waters.

9 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. A. D. Viney and J. Gray were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows :—

AVENT, ALEC, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanider. *(Transfer from Associate Roll.)*

BLAU, MAX, Mynbouw Maatschappij Siman, Lebong Tandai, *via* Benkoelen, Sumatra. Mining Engineer.

HILLIARD, CECIL BREWITT, Robinson Central Deep, Ltd., P. O. Box 1145, Johannesburg. Cyanide Manager. *(Transfer from Associate Roll.)*

KEWLEY, ROBERT RIGBY, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Sampler.

PHILLIPS, JOSIAH CHARLES, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Cyanider.

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting :

ANDERSON, ROBERT, East Rand Proprietary Mines, Ltd., P. O. Box 30, East Rand. Tube Miller.

EASTEN, GORDON TRAVERS, East Rand Proprietary Mines, Ltd., P. O. Box 94, East Rand. Tube Mill Foreman, Cason Battery.

INGRAM, JOHN, Durban Roodepoort G. M. Co., Ltd., P. O. Box 111, Roodepoort. Cyanider.

MACFARLANE, WALTER GRÆME, Jumbo Mine, Mazoe, Rhodesia. Assayer.

NICHOLAS, DAN, Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg. Assayer.

O'CONNOR, BERNARD, P. O. Box 6510, Johannesburg. Cyanider.

RICHARDSON, WILLIE PARK, Forbe's Reef, near Mbabane, Swaziland. Amalgamator.

WACHER, HARRY, Jumbo Mine, Mazoe, Rhodesia. Cyanider.

WOODS, WALLACE, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Tube Mill Foreman, Driefontein Battery.

GENERAL BUSINESS.

The Secretary read a letter from the "Metallurgical Trials' Committee" stating that in future the name of the Committee would be "The Mines Trials' Committee."

The President : I think that this change in the name of the Trials' Committee is a very good one, because it shows that the scope of the trials has been very greatly extended. The Mines Trials comprise everything connected with mining, whilst the Metallurgical Trials' Commitee practically confined themselves to metallurgical matters. The Mines Trials is an exceedingly broad term, and will comprise rock drills and all other appliances connected with mining, and any inventions or improvements in mining work and attendant metallurgical processes which anyone may have to bring forward. I only hope that any of our members, who may have in their minds any improvements or inventions, with regard to mining or metallurgical work, will bring these before the Mines Trials' Committee, and I am quite sure that they will meet with a fair consideration and a fair trial.

REMINISCENCES OF THE EARLY RAND.

By M. H. Coombe (Member).

In the early part of 1887 I arrived on the Rand from Kimberley, anxious to try the new Eldorado and see if I could not wrest from it that fortune I was eagerly looking for.

I came up in a four-wheeled covered spring wagon drawn by six mules, which in addition to its eight passengers, owner and driver, carried as freight a section of a flywheel. I mention this as it tends to give one an idea of the difficulties of machinery transport in those early days. The old engines, mills, boilers, and machinery generally, were transported hither in all manner of conveyances and in sections and parts to fit the vehicle that was to convey them. Broken down wagons, carts, and stranded machinery, dotted the side of the road all the distance between here and Kimberley, which was then the rail terminus from the Cape Colony side. The marvel was how all the widely parted pieces of the mill or engine ever arrived together in one place and became united into the perfect running machine. However, the flywheel section with the wagon and its passengers duly arrived and outspanned on the Market Square, just then in the process of having its fringe of buildings reared around it. Work appeared to be concentrated on the southern side of the square, where a long line of wood and iron buildings (wood lined) were being erected to serve as offices, cafés, etc., men taking contracts at long prices and working day and night at feverish heat to get them finished and join in the run for more. The result was that in a short time houses sprang up like mushrooms, and a city throbbing with life and energy was evolved from the veld. There was no long line of headgears and smokestacks nor throbbing sound of stamps then to guide the stranger to the reef outcrops. I enquired the direction and set off to the Wemmer, that being the nearest mine on which work that could be called mining was in progress. Arrived there, I found an open cutting probably 200 ft. long on the outcrop strike and 40 to 50 ft. deep, at the bottom of which a mixed assortment of natives, Cape boys, and Europeans were at work, drilling and filling ore into buckets, which in one instance were hoisted with a whip, oxen being the motive force, and in the others with windlasses, white men being on the handles.

About 100 ft. west of the western end of the open cast a little 6 ft. × 4 ft. shaft was sunk from which a drive was proceeding towards the cutting, all the work being done by white men. I do not remember seeing the slightest evidence of machinery on the Wemmer at this time, if I except the whip and the windlasses, and I think they were trying to keep a mill going out of the cutting. Such was the genesis of mining on the Rand. I enquired for the manager, and he was pointed out to me, an elderly man standing with his hands in his pockets gazing down into the cutting. Approaching him, I said "Good morning," and asked him if he had a vacancy for a miner? He said that he had, and that the sight of a miner would do his eyesight good, as he had not seen one in the country yet, and expressed his opinion that I was probably like the rest of the so-called miners of this country. When I quoted De Beers as the last mine on which I had worked, Captain Wells waxed very sarcastic, but when, seeing how the land lay, I casually mentioned that I had been brought up on a New Zealand gold field, the old chap, who was an Australian, after several questions offered me work. The job was to cut hitches and put in a stull in the open cutting 10 ft. above where the men were working. The hanging was showing signs of weakness, and he intended to stull and fill in to surface. I and another miner were to do the work ourselves, no "boys" being available. I asked very quietly what the contract price was, and Captain Wells gave me plainly to understand that there was no contract in the matter at all, that it was a question of days pay, the ruling rate for which was £2 5s. a week, but as this was a special job he would give us an extra 5s. and make it £2 10s. a week. This was the first intimation I had of the rates of pay on the Rand, and after a glance around on the sky and veld to make sure I was wide awake, I meekly enquired if he had not made a mistake in saying £2 10s. a week! did he not mean £2 10s. a shift? I draw a veil over what ensued; suffice it to say that the manager exploded, and we retired under a running volley of remarks that would have wiped out an army corps. The next mine east where any work was going on was what is now the Salisbury G. M. Co. Here there was the usual cutting in line on the outcrop with a few white men and natives at work, drilling, shovelling, etc. There was also a cross-cut running north at right angles to the cutting, and in this I could have started work at the usual rate of 45s. a week. One incident occurred here I would like to mention. Coming back to the cutting—it would then be about noon—I saw a man come out of the cross-cut unrolling a coil of fuse. Turning the corner into the cutting he unrolled the coil to the end and then proceeded carefully to build a small heap of dry grass round it. He shouted to me in Dutch to go quick as he was about to fire. I watched him light the grass and saw the fuse "spit" through the flame, and as the man hastily scrambled out of the trench I walked slowly away and was away down near the Natal Spruit when I heard the report of the hole. That was undoubtedly a sample of mine efficiency and economy in the early days. I have somewhat enlarged on the experiences of my first day, because I want to show that the Rand was originally, in the main, opened up by white labour. It was a bit of a shock to a man who had been accustomed to anything from £10 to £20 a week contracting in De Beers to be offered 45s. and 50s. a week, but that was the ruling wage for white men on the Rand in 1887, and this was not for bossing up Kafirs, but for hammer and drill work and manual labour generally. As Kafir labour became more plentiful and miners came to the Rand in greater numbers, the present system of an aristocratic white labour section and a native unskilled labour section became evolved.

After a day or two spent in Ferreira's camp studying the situation, I decided to journey out west and interview an old friend of mine who was in charge of the 10-stamp mill at the Kimberley Roodepoort. I remembered once accompanying him round Kimberley, visiting the chemists' shops and making enquiries for cyanide of potassium, and I believe the quantity he required had to be made up by sending to Port Elizabeth. The Kimberley Roodepoort was the only floated company working in that district at the time, if I except the "Evelyn" adjoining it to the west, on which a vertical shaft, located to the north or footwall side of the outcrop, was being started to be sunk to a depth of 300 ft. This monument to the sagacity of the then Rand mining expert was afterwards filled in. The Kimberley Roodepoort 10-stamp mill was the only one at that time west of Johannesburg, excepting Struben's, that was actually milling, and great things were expected of it. My friend, the millman, told me that the disappointment would be greater than the expectation, and so it was, and always has been with this particular property. I was present at the first month's clean-up, and saw the first act in the drama of disappointment played. The clean-up was duly placed in the old fashioned open air retort, the fire carefully attended to and the blanketing on the pipe nozzle of the retort kept wet. When evaporation had ceased the retort was lifted off the fire, the cover removed, and to the eager expectant eyes was revealed a round blackish ball of sponge gold not much larger than a cricket ball. It was carefully taken out and washed and then the fun began. The manager balanced it in his hands guessing at the weight, it was passed to the millman for his guess of weight, and slowly it began to dawn in the manager's mind that the result of the month's crushing was far, very far below the estimate of his friends and supporters. Then he began to say hard things and cast reflections on the honesty of all concerned in the production, which in turn set the millman off at a tangent, who, being a man of parts with a few years education in the flowery language of the Western States mining camps, soon convinced the manager that he did not possess a monopoly in the use of sarcasm and scathing epithets, and ended his oration by throwing the ball of gold as far as he could across the veld and telling the manager to go and get it if he wanted it!

Across the valley from Roodepoort on the rising ground to the north a company had been floated and registered as the Branstone G. M. Co. They erected a 15-stamp mill, all steel framing built in Natal, but they never crushed an ounce of ore from the Branstone. I obtained a contract from the manager to sink two shafts and open the mine out for the mill. There was a small poor looking banket reef on the contact between the quartzites to the south and the Hospital Hill shales to the north. The shafts were about 100 ft. apart and were to go 40 ft. deep and be connected. No particular size or shape was stipulated; my only instructions were to get down and get through. Shortly after starting to work, the mill was finished and had to be run, so a number of Scotch carts, wagons, etc., were engaged to ride quartz from the waterfall. This was actually done for about three months when the mine closed down for re-construction. It has never worked since, and but few people know that such a concern ever existed. I was engaged shortly afterwards at the rate of 50s. a week with a possibility of a contract, to open up a mine for a company on whose property a mill was being erected. This was the Aurora G. M. Co. I collected some materials, tools, and eight very dilapidated looking natives, and when I was ready a wagon was hired and I set off to find the property of the Aurora G. M. Co. The only directions I had were to proceed to Paardeberg, and I would find the beacons on the hill to the south; I was also told to start mining operations at once, the plan and scope of such operations to be decided by myself. I was a miner and I should know. I found the reef outcrop by the usual trenches and cuttings which had been put in it and across it by the prospectors, and traced it through the property. I then located all the beacons, made a rough sketch of the boundaries and proceeded to locate a shaft. I found a small prospecting winze sunk on the main reef about 30 ft. deep, with a vertical hole sunk through from the surface to the bottom of the incline. I decided to square up, widen, and sink the incline, and started the boys on this work.

I had no windlass, no buckets, no rails, no timber, so the ground had to be "shambled" out. Weekly pay was the rule then for black and white, and all boys were paid alike, viz., 20s. a week. When Saturday came I had to journey to town to draw the weekly wages for the boys and myself, and all the time, about eight months, that I ran the work, I was my own secretary, timekeeper, and paymaster. At the time I left there were six white men and 200 boys working, so that in 1887 and 1888 a man in charge had many and various responsibilities attaching to his office. I came to town every Saturday morning, received a cheque which I presented at the bank, and went back to the mine with the cash, paying the men and boys that afternoon. In this way I received the first Queen's Jubilee coin paid over the Standard Bank counter in Johannesburg—at least the clerk told me so when he paid it out.

We sank the shaft to a depth of 85 ft., which brought us to the water level, so we started driving east and west, and as the mill, three miles from the mine to the south, was nearing completion, when the drives were in about 30 ft. each way we started stoping. Our plant, at the time the 10-stamp mill started to crush ore, consisted of one windlass, two wooden skips, each about 2 cub. ft. capacity, no trucks and no rails. We were using yellow wood 4 in. × 4 in. scantling in lieu of rails for our shaft track, and wheelbarrows in our drives. The Scotch carts for ore conveyance to the mill were, of course, hired. The same conditions of work obtained on the Nabob and the Edinburgh, two properties immediately adjoining the Aurora to the east. I soon pointed out how impossible it was to go on under these conditions and put in a requisition for an engine, boilers, pumps, rails, trucks, timber for bins and headgear, etc. This had for effect the appointment of an engineer, one of the most hopeless "experts" it has been my misfortune to encounter. He was a German railway engineer and, worst of all, had been an officer in the German army during the Franco-Prussian war. This latter qualification came very near being our undoing, so far as working the mine was concerned, because the men got to know his weakness and that nothing pleased him better than to get them around to hear him fight his battles o'er again. I have known him have all the men off work a whole afternoon listening to these recitals. This gentleman's mining knowledge was very limited, as the following example will show. Driving east about 30 ft. from the shaft we came up against a fault. In an open cutting on the surface this fault could be plainly seen and the throw determined, the reef being faulted 8 ft. south. Our engineer was very much perturbed about this, and after a minute examination underground, he drew up an exhaustive geological report proving conclusively that we had reached the limits of the vein east. I told him we could pick up the reef by crosscutting south.

We received at this time a visit from an Australian expert, specially imported, I believe, by the Transvaal Government, to report on the banket reefs of the Witwatersrand, and ours being one of the deepest shafts, he came there to make some investigations. He had no faith in the future of these fields, and gave as his dictum that it was an alluvial wash and would peter out or disappear at about 300 ft. vertical depth, at any rate, if the bed continued the gold values would cease to exist. He is not the only expert to whom the auriferous beds of the Transvaal have been like Cape Colony to its Governors, the grave of reputations. The old days of the Rand appear as a dream or as days of childhood to the grown man. We did not think of the mighty edifice that was to be reared on the foundations we were building. We saw the birth and sturdy youth of our mining industry, have we seen its prime, and shall we or our children see its old age? We could not prophesy in 1887 of the greatness it would reach in twenty years, so what living man dare prophesy of its magnitude at the end of the next twenty?

And let me say here that we owe but little of our progress to knowledge brought from other mining centres. We have evolved our own practice in mining and metallurgy and stand to-day in the position of leaders and teachers to other fields. In the days I write of it, was the common practice to take the ore to the water, sometimes over long distances by cart and wagon. For instance, the Ferreira mill was out Auckland Park way. The old Mint mill still stands at the bottom of Fordsburg hill, and the Natal Spruit below the City and Suburban was thickly studded with small mills. The lesson was eventually learnt to take the mill and water to the ore! I think the Spes Bona was the first to adopt this practice as it was also about the first to instal the electric light. It is a big advance from the 1, 2, and 3 compartment shafts of 1887, 1888, and 1889, with their 4 in. × 4 in. yellow wood slides, and their single cages and 10 cwt. skips to the 5 and 7 compartment shafts of the present day equipped with steel guides and 4 and 5 ton ore skips. It is also a big advance from the days when the mill tailings were allowed to run into the sluit, and the mine manager prayed for nothing better than a good storm of rain to wash them away so that he might have room for more, to the cyanide works and slimes dams and plant of to-day.

In every department of mining vast strides have been made, and we have borrowed nothing, but have evolved and created our own processes and our own system and style of mining. Even the miner from other fields has to practically learn his business over again in these mines. Our footage in sinking and development work is far ahead of any other district in the world, and taking the size of the reefs, more tonnage is stoped here by one man in one day than is in most countries in two. There has been a lot of idle talk about the inefficiency of our miners, but I hold that, as a class, they are the best of the world's mining camps, and to them is due much of the credit for the magnificent gold industry we see around us, some of the humble beginnings of which I have endeavoured to illustrate to you.

The President: Our thanks are due to Mr. Coombe as a very old resident and pioneer of these fields, for reading us this most interesting paper. I consider myself an old pioneer of the Rand, but Mr. Coombe was a very little bit before me. Of very many of the incidents which he has related in his paper I am also personally cognisant. He mentioned that when he arrived on the Rand he went to the Wemmer G. M. Co. and found Captain Wells standing on the top of an open cutting surveying the work. I have in my possession a very old photograph, of which I have only seen one other copy in a shop window where I bought my photograph many years ago. It shows that open cutting of the Wemmer, and Captain Wells standing upon the top of the earth thrown out from the cutting. I shall be very pleased to bring that photograph and to show it to you at the next meeting. It is a most interesting one, showing old Johannesburg, the only double storied building being Henwood's on the north side of the Market Square. With regard to the author's remark concerning the old practice of bringing the ore down to the mills which were then erected very close to the water, when I came out here originally, the first work I obtained was on the West Rand, and I used to pass a certain property very frequently. It is not now working, but I hope will shortly commence again. A mill was then being erected close by the spruit. I used occasionally to have a chat with the gentleman who was in charge of the construction work, and one day I ventured to suggest to him that I thought it would have been better to put the mill on the slope and pump the water up to it. He regarded me in silence, as if he could find no language to express what he thought of me or my capabilities, and I knew I was looked upon as a fool. A few years later I was working in the same mill and I know what it was. We did pray for a good shower of rain to come and wash the tailings away, because, as a rule, after a few days' work, the tailings would get banked up until at last they rose to the foot of the tables, and we had to stop the mill and turn the boys out to cut a trench to try and induce them to run away. When one looks back upon those old days, one feels a certain amount of amusement when considering what has been evolved from those conditions to those at present. One also looks back with a great amount of interest, for as Mr. Coombe says, the whole thing has been evolved out of the Rand itself. As I mentioned in my inaugural address, we have certainly borrowed the highest talent that was available, but still I take it that when that talent was obtained, it became Rand talent, and remained so. I think all those who had any connection with the Rand in those early days and with the work of bringing mining to its present state, are proud of their connection with it. Mr. Coombe is to be congratulated in having put these reminiscences before us in a most interesting and instructive manner.

RESCUE APPLIANCES: LESSONS FROM GLENCOE.

(*Read at March Meeting, 1908*).

By H. Kestner.

DISCUSSION.

Mr. A. Heymann (*Member*): The difficulty I have in making any remarks on the question of safety appliances connected with mining is that these matters are being considered by a certain Commission to which I have the honour to belong, but I think there are a few general remarks which I may make and which cannot be considered a breach of faith. No one will deny the necessity of having ready at hand some kind of reviving or safety appliance, though on these fields we are in the fortunate position of not being often in need of such. Our coal mines, for instance, cannot be classed as "fiery mines," as most coal mines are. There are very few of them which have noxious gases or where gassing happens, and still less are our metalliferous mines liable to that kind of thing. But we do sometimes have gases derived from explosives which cause serious, and sometimes fatal, accidents. If we are to be in a position to save even one life we must be provided with some kind of rescue apparatus. In choosing an apparatus, I should certainly say, choose the very best one obtainable. There are many types known, and they vary in their efficiency and comfort of wearing. I do not care to express any opinion as to which is the most suitable apparatus. It is not sufficient that rescue appliances should be provided, but men should be trained to use them, so that when occasion arises they will know exactly what to do. I have had occasion to use several kinds of apparatus. I was conducting a certain experiment in a mine where it was necessary for me to collect the gases immediately after an explosion. I was wearing a helmet, and directly after the detonation had taken place I rushed to the place in order to get my sample of gas. The helmet was a huge thing, in which were glasses for the eyes, but I could only see directly in front of me. In order to see when stooping I had to bend double, so it can be imagined how uncomfortable I felt. I fell at every few steps. On coming back to the station, and before I was out of the danger zone, I could not stand it any longer, and took the helmet off as I could not breathe comfortably with it any longer. Perhaps it was my own fault, as I was not used to it. On another occasion I had another kind of apparatus supplied with air, which was supposed to last 40 to 45 minutes. I used it for only about 20 minutes and thought it was quite safe, but after using it another 5 minutes I found that I could hardly breathe. I had also occasion to try a Shamrock. It may have suited the particular work which I had to do, but it also satisfied me. I could run with it, I did not feel any weight, and it was very comfortable. Another advantage I should say to such a type of apparatus would be ; suppose a case of fire in a coal mine, and one has to work for some time in a stationary position, it can be attached to the compressed air pipes and a supply of air can be obtained from the compressor. This I consider is a great step forward. There may be other apparatus just as good or better, but I have not seen them. Turning to reviving apparatus, there is one invented by Dr. Bratty which is very ingenious, in that it imitates the natural breathing very closely, and by which, when a man is gassed he can be given a certain amount of oxygen which can be easily regulated. Such an apparatus as this would be most useful in cases of gassing. I can assure you that if one had been in danger or even if not in danger oneself, he had had occasion to see the horrors which occurred at such an awful accident as, for instance, that of Glencoe, which I can still see before my eyes, no matter what the expense would be of obtaining the necessary rescue and reviving appliances, we should all strive, even if the cost came out of our own pockets, to have them at our disposal whenever required. I may say that the Mining Regulations Commission, to which I have the honour to belong, has had under consideration several of the best types of apparatus which a certain medical gentleman was deputed to send out from Home, and their efficiency is being considered by the Commission with a view to recommending the best. Not only efficiency but uniformity is necessary in such an apparatus, so that the miners and others trained in their use should be able to use them in emergency on any mine. I regret to see that the paper submitted by Mr. Kestner did not evoke much discussion. I should also like to have seen the appliances of other firms so as to form an opinion as to which was the best.

The President : We are pleased to hear Mr. Heymann's remarks on this subject. He has had a great deal of experience in the matter of noxious gases, and I must agree with his remarks as to the necessity for having some approximation to a standard type of apparatus, and the training of men in the use of the same. I think that training plays a great part in connection with these appliances, and I should also include the fire apparatus in the same category. I recollect a man being required to lay out 200 ft. of hose and join on to that another 100 ft. To a man who knows how to do this there is nothing easier, but this man got himself most hopelessly tied up in the thing because he fastened it at the wrong end and tried to pull the hose out from the centre, and it took him a very long time to untwist the hose and get it right. Another man got a similar hose out as fast as he could run simply because he knew how to do it. If the first man had had to deal with fire, I should have been very sorry for the building concerned.

Mr. A. Heymann : I understand that Prof. Yates is connected officially with the St. Johns' Ambulance Association. May I suggest that he should arrange for the curriculum for the First Aid classes to include training for gassing accidents and practice with rescue appliances such as we have been discussing.

Prof. J. Yates (*Past-President*): I will look into the matter.

REPLY TO DISCUSSION.

Mr. H. Kestner : In replying to the criticism on my paper, I want first of all to add a few details, and then to reply to the only critic, Mr. Tom Johnson.

As you will have seen from the report of the English Royal Commission,* the absorption of CO_2 in the "Shamrock" apparatus has been termed "bad": but in the tests carried out by the above Commission the old "Shamrock" type —known as the "Breast type"—was employed. Furthermore, it must be taken into consideration that the apparatus in question was used without any instructions from the makers, and therefore it is not certain that the apparatus was in good working order.

---

* *South African Mining Journal*, March 21, 1908.

As I stated, the "Westphalia" type is a great improvement on the older one, and the makers have discarded the name of "Shamrock," calling it at present "The Westphalia." The latest tests made with this apparatus,* prove that it has a CO_2 absorption as good as can be desired. I will return to these tests later on, but I might also mention here that similar tests were made for the Mining Association of Bochum, which also show the excellent working of the new type. These published tests have not by any means been "show tests" as they were carried out independently by the Mining School at Bochum. The Lancashire and Cheshire Rescue Committee in England have also made very conclusive tests with rescue apparata, in which the Westphalia apparatus has always turned out the superior.

It has specially been shown by medical facts that the persons operating with this apparatus have been in excellent condition after the test. Further, the work has been done by persons wearing it in 20 minutes less time than by those wearing other apparata; and this result has been attained in spite of the fact that the wearers had not any previous knowledge of the apparatus, whereas, on the other hand, the wearers of the other types, amongst these the "Dräger" type, had been thoroughly accustomed to and experienced with their several types.

The Royal Commission further says that it is a great advantage for any rescue apparatus to have the gauge visible in front of the wearer; but a practical demand for this arrangement does not exist, and this statement must be accredited only to local patriotism. The "Weg" apparatus of Mr. W. E. Garforth has such an arrangement in front, but everyone who has studied the literature dealing with the progress of rescue apparata will find that the advantages claimed by Mr. Garforth have been known for many years, and consist of arrangements which have never proved their reliability. It must be recognised that he has constructed the "Weg" apparatus for the benefit of mining communities, and as he has a great deal of influence the popularity of the visible gauge can partly be attributed to this. If the visible gauge has been adopted for the "Fluess" apparatus, it can only be regarded as a certain concession to the popularity in England of this arrangement, which is a great danger, as a long pipe line is required, which, under high pressure of 120 atmospheres = 1,720 lb. per sq. in. is very difficult to construct, apart from the fact that this danger does not compensate for the visibility of the gauge. Really there is no need for the gauge to be visible to the wearer. Different accidents have proved that in very serious cases it is not advisable to send a man single-handed into the gas-laden atmosphere, but a rescue party should go forward in companies of at least four or five men, and in this case each man can control the gauge of his fellow-explorer. As mentioned, the "Shamrock" apparatus can have a signalling device attached, and I wish to mention here that at the time of writing I had been misinformed about the signalling device, and said that it was regulated by an electric contact; however, in the meantime, I have received fuller information and show this alarm in detail in Figs. 1, 2 and 3. This device can be fixed on the pipe leading the compressed oxygen to the reducing valve, and it can be so regulated that as long as the gas pressure remains at 30 atmospheres = 430 lb., that is, as long as the apparatus contains sufficient oxygen for half-an-hour's work, the whistle is closed and the valve *a* (see Fig. 1) is pressed closely on the seat. If the pressure goes below 30 atmospheres the spring *c* extends and moves the valve cone *b*—thereby also the valve *a*—so far that the oxygen passes through the alarm whistle, as shown in Fig. 3. The whistle sounds

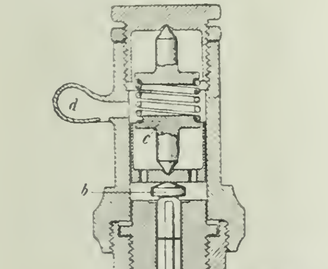


FIG I.

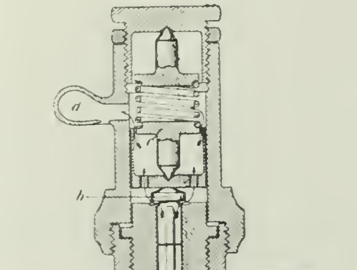


FIG. II.

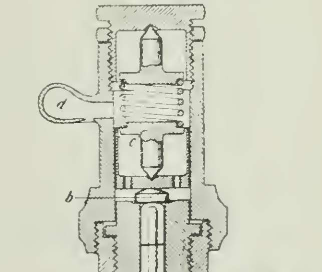


FIG. III.

---

* *Glückauf*, No. 27, July 6, 1907.

until the gas pressure is below 20 atmospheres = 290 lb., and then spring *c* again presses the valve cone *b* tightly on the seat so that the outlet for oxygen is closed again. This position is shown in Fig. 3.

The following are the results of tests made by Mining Director Grahn in company with Mining Chemist Dr. Dieckhoff and Mining Diver Korte. The tests made were on three different days with the same two apparata, and are as follows :—

The absorption is thus quite sufficient, and the different mining students who executed the tests were in splendid condition after completion, and performed the above-mentioned work without difficulty. It is not to be wondered at, that at the end of the two hours trial, the air contained a high percentage of CO_2, as it is natural that the regenerating substances should be nearly used up. The air circulating in the apparatus was about 50—75 litres = 11—16¾ gallon per minute

Date.	Time of Test.	Inhaled Air contained CO_2 per cent.	Work Done Measured.	Remarks.
31 May, 1907	11.15 a.m. to 1.15 p.m.	at 11.45 a.m. = 0·36%	= 8,437·5 mkg	Condition of wearer : Good.
		12.15 p.m. = 0·80	= 8,437·5 ,,	
		12.45 p.m. = 1·02	= 8,437·5 ,,	
		1.15 p.m. = 2·40	= 10,125· ,,	
		Total	35,437·5 mkg	
,,	11.25 a.m. to 1.25 p.m.	at 11.55 a.m. = 0·34%	= 8,437·5 mkg	Condition of wearer : Good.
		12.25 p.m. = 0·08	= 9,000· ,,	
		12.55 p.m. = 0·26	= 11,250· ,,	
		1 25 p.m. = 0·22	= 11,250· ,,	
		Total	39,937·5 mkg	
1 June, 1907	9.15 a.m. to 11.15 a.m.	at 9.45 a.m. = 0·20%	= 7,312·5 mkg	This Test cancelled, as one connection was not tight and admitted atmospheric air. Condition of wearer : Good.
		10.15 a.m. = 0.04	= 9,000· ,,	
		10.45 a.m. = 0.50	= 9,000· ,,	
		11.15 a.m. = 1.48	= 8,015·625 ,,	
		Total	33,328·125 mkg	
,,	9.18 a.m. to 11.18 a.m.	at 9.45 a.m. = 0·06%	= 8.437·5 mkg	Condition of wearer : Good.
		10.15 a.m. = 0·08	= 9,000· ,,	
		10.45 a.m. = 0·20	= 9,000· ,,	
		11.15 a.m. = 0·46	= 8,296·875 ,,	
		Total	34,734·375 mkg	
7 June, 1907	11.5 a.m. to 1.5 p.m.	at 11.35 a.m. = 0·36%	= 7,593 75 mkg	Condition of wearer : Good.
		12.05 p.m. = 0·38	= 9,281·25 ,,	
		12·35 p.m. = 0·62	= 8,437·5 ,,	
		1·05 p.m. = 1·66	= 9,281·25 ,,	
		Total	34,593·75 mkg	
,,	11.8 a.m. to 1.8 p.m.	at 11.35 a.m. = 0·32%	= 8,015·625 mkg	Condition of wearer : Good.
		12.05 p.m. = 0·18	= 8,015·625 ,,	
		12·35 p.m. = 0 36	= 7,171·885 ,,	
		1·05 p.m. = 1·30	= 8,437·5 ,,	
		Total	31,640·635 mkg	

at the suction head of the injector of 8—11 cm. $= 3\frac{1}{8}$ in. to $4\frac{3}{8}$ in. water column.

For these tests the same two apparata were used, but by different mining students.

In replying to the criticism of Mr. Tom Johnson, I very much regret that—probably on account of want of space in the *Journal*—my remarks regarding the oldest types of rescue apparata were omitted, otherwise it would have been seen that I am quite aware of the existence of the apparatus mentioned.

He speaks especially of rescuing gassed persons in gold mines, but I think he will admit that this is not such a serious matter as to require special rescue stations being fitted out. Dr. Bratt's patent reviving apparatus is one especially adaptable for reviving gassed miners ; it has also a great advantage in that it is easily worked, and any intelligent Kafir—after a little explanation—would be in a position to handle it.

He states that he came to the conclusion that smoke helmets and bellows are as good as most rescue appliances, but this, I believe, he will not say seriously. My paper was given with special reference to the accident at Glencoe, and it has never entered my head to say that no smoke helmets should exist ! I think that the smoke helmet is a very useful device for fire brigades or in case of gassing in gold mines, but I maintain —and hope Mr. Johnson will agree—that in cases of accidents like the Glencoe disaster, the smoke helmet is quite useless.

He also refers to the Aerolith. I have nothing special against this apparatus, but its arrangement forbids its extensive use in coal mines. The advantage of the Aerolith in that, if men using it have to stay at a fire or in gas for a longer time than the supply lasts, fresh supplies can be emptied into the knapsack without the wearer having to take it off, is naturally also claimed for the circulation apparata, such as the Dräger and Westphalia, or any other type, and the greatest disadvantage is that it cannot be constructed without a valve, and a valve is always liable to breakdowns.

I regret the omission in my paper regarding the Dräger apparatus worn by the Paris firemen in the Courrières disaster. Mr. Johnson is right in stating that their rescue party wore this apparatus. I beg to say here, that I think no serious mining men at home would consider any but an oxygen circulation apparatus.

Mr. Johnson says that this type of apparatus has no lives to its credit yet, but I beg to differ, and can give instances where, in the Westphalia District, the oxygen circulation apparatus has saved many lives. There is another point upon which I differ from him, and that is, his proposal to fit up a rescue station with five types of apparata. I think such a station quite impossible. Firstly, every man employed in rescue work must be, without doubt, experienced in and trained with the apparatus ; whether he be a fireman or a miner does not matter as long as he is possessed of the necessary experience, knows how to handle the apparatus, how to behave in time of danger, and has a certain amount of pluck in the event of any unforseen occurrences ; but, I admit that he is right in saying that every mining student of the Transvaal University College should have a full knowledge of the best methods in rescue work.

I am informed that the Mine Managers' Association of Natal has planned the erection of a central rescue station and the employment of a trained rescue brigade, and I congratulate the association on this foresight and hope that—should such a station come into existence—it will prove of great value to all the Natal mines in case of accidents.

I should like to point out that it has not been my intention to "push" any special apparatus, but as I have had a certain experience—firstly, as a born Westphalian, and secondly, as a man interested in rescue appliances—I consider, and still maintain, that the "Westphalia" is one of the best.

In conclusion, I beg to tender my heartiest thanks to Mr. Heymann for his valuable remarks on rescue work, and fully endorse his words. I am glad that he comes to the same conclusion as I, viz.: that oxygen circulation apparata are the only reliable ones.

The President : We must congratulate Mr. Kestner on his excellent reply to the criticisms on his paper, and I hope that when the improved Westphalian apparatus is brought out here, he will be able to induce the agent to give a demonstration of the apparatus before the Society. I think it is the rule that we get gassing accidents most unexpectedly. Men and boys will be working in a certain place day after day and nothing happens, and then suddenly both men and boys are gassed. Usually they have to be brought to the surface. Why not have some appliance below, to deal with them as quickly as possible, because in cases of gassing, the longer a man remains under the influence of the noxious gases, the greater the difficulty of reviving him. With the possibility of doing something at once, recovery becomes much more possible. I am quite sure many a life is lost for the want of the immediate application of some life reviving appliance. I think it is a matter which ought to receive greater consideration on the mines, and it should not be looked upon as a matter of expense. It is a matter of saving life, and life is valuable, whether it be a black or white man. The cost of suitable appliances spread over the large tonnage now being yearly handled by most of our mines would form a very small item in the working costs.

Mr. H. Kestner: As soon as the apparatus arrives I will endeavour to have it brought before the Society.

A FEW NOTES ON STAMP MILLING.

(Read at April Meeting, 1908.)

By W. H. Jane (Associate).

DISCUSSION.

Mr. J. Kennedy (*Member*): In discussing this paper I hope my belated contribution will be of some interest to the author and to the members of this Society generally. I had hoped that some of the older members would have come forward, and given us some information regarding the points raised by the author as to the five stamp cycle theory, etc.

As I have recently come into possession of some data dealing with a five stamp versus a four stamp battery (for which I am indebted to Mr. Howard J. Lee, battery manager of the North Randfontein, where the tests were carried out), I venture to hope that the following results may be of interest to the members :—

Experiments on North Randfontein G. M. Co. four stamps against five. Before giving the results of the two tests carried out on the above property I should like to state that the tonnage was as accurate as could possibly be obtained under these circumstances, *i.e.*, by trucks (the fine and large rock being weighed and an average taken). Apart from this, every precaution was taken to ensure a fair test. New stems, heads, tappets, shoes and dies were placed in both boxes (boxes being of the same make and having been given the same discharge). The shaft for the four cams was specially drilled to allow of a regular drop, in other words, so divided as to ensure eight regular drops per revolution (no pause between any two stamps). The order of drop being 1, 3, 2, 4.

Result of First Experiment.—Four stamps crushed 96·8 tons in 3·468 days. Duty per stamp per 24 hours—6·971. Five stamps crushed 96·8 tons in 2·766 days. Duty per stamp per 24 hours—6·999 (a difference of ·021 in favour of 5 stamps).

Result of Second Experiment.—Four stamps crushed 102·96 tons in 3·730 days. Duty per stamp per 24 hours—6·900 tons. Five stamps crushed 120·12 tons in 3·481 days. Duty per stamp per 24 hours—6·901 tons (a difference of ·001 in favour of five stamps. Screening used 900 mesh. I should like to add that during these experiments two men working 12 hour shifts did nothing else except look after these two batteries.

From the foregoing tests it will be seen that the cycle theory does not amount to much, although the difference is in favour of the five stamp battery, which is probably due to the increased agitation in the mortar caused by the extra stamp. I may mention that in the case of the four stamps, one end of the mortar was filled in with wood and lined with iron plate so as to avoid waste space.

With reference to the author's theory of running five stamps in one battery to obtain a steady load on the driving belt, I should like to know what is to prevent one from having 6, or even 4 cams, as was done in the above test, arranged on a shaft in order to ensure a steady pull on the belt.

The author repeatedly contradicts himself in first suggesting that, Nos. 1 and 5 stamps should have a longer drop than Nos. 2, 3, and 4; secondly, that the centre stamp should have slightly heavier shoes or a slightly longer drop than the others, and then adds that all stamps should be run at the longest possible drop. Now, if one stamp has more drop than another in a battery, it is very evident that they are not all running at the longest possible drop. I quite agree with the author that drops should not be shortened more than the maximum amount of wear of the shoes and dies per day, at any one time, but, in a 300 stamp mill such a system is hardly practicable, The idea of cautioning mill men against the practice of shortening drops by inches at a time is good; but surely no man with the slightest pretence of being a mill man is indulging in that practice on the Rand to-day.

The author asserts that stamps running 98 drops per minute, with a maximum drop will do more work than stamps running at 100 drops per minute, because of the increased drop in the former. Well, I prefer the latter for the reasons that the difference in length of drop between the two is so infinitesimal, and that the additional 10 drops per minute in the latter create an increased agitation in the mortars and so help to expel the crushed ore expeditiously. If the author really favours the running of a longer, but slower drop, I might refer him to Gilpin County, Colorado, where some years ago it was customary to run stamps with from 16 in. to 18 in. drop at a speed of 26 to 28 drops per minute. Just how they are now being run, I do not know, but I expect they are more up to date by this.

With regard to making stamps do too much work, I have never heard of such a thing; on the contrary, my experience has always been that one can never crush enough to satisfy the management.

REPLY TO DISCUSSION.

Mr. W. H. Jane (*Associate*): I must express regret that my paper has not brought forth more expressions of opinion regarding stamp milling practice. When we consider the enormous strides our mechanical engineers have made in design and construction of stamp mills, I think the running staff ought to make greater efforts to profit by the improvements made. Two heads are better than one, and by free expression of opinion, we can do much to further our object. I have discovered, during my milling life, that it is sometimes possible to learn from a learner, and get out of some of the ruts. How much more can we learn from each other? I thank our President for his kindly consideration. In regard to the order of drop I have not had much chance to experiment, but I believe I got a more even discharge from the boxes by the following 1, 3, 5, 4, 2, than with the drop mentioned, 1, 4, 2, 5, 3. This drop was installed throughout the mill, and on changing a cam shaft I replaced it with the first named order. The tappets were set perfectly on this drop, but on the new drop I had to shorten Nos. 1 and 5 stamps, $\frac{1}{2}$ in., and Nos. 3 and 4, $\frac{1}{4}$ in., making No. 3 the longest drop in the battery, and Nos. 3 and 4, $\frac{1}{4}$ in. longer than Nos. 1 and 5. However, I have no data to prove that we did more or better work. Regarding the number of drops per minute, I came to the conclusion, after running about two months at 102 drops per minute, that we were not doing all we could do, and slowed down to 99 drops, when I found we were doing more work with less wear on the mill. This was on Nigel ores, which are very hard. The breakers, Blake-Marsden type, were under my own control, and I ran them with the jaws touching whilst running, and they were screwed up about every third day. The ore was broken to less than 1 in. cubes, and only when the jaws wanted changing was the ore allowed to become larger, a rare occurrence, but beneficial, inasmuch as it demonstrated beyond doubt that it costs less to crush rock in the breakers than by means of stamps. The new type of feeder drive installed at the Simmer Deep gives promise of a big improvement, except that the objectionable spring is in evidence. I hoped that my question *re* tube mills in conjunction with stamp mills would have given us a standard to work from. What I wish to know is what percentage of ·01 may we retain? I know that on assaying sand residues, which had not been classified, as no tube mills were installed, we found the high values in the ·01 product, but by carefully classifying the mill product in classifiers will not the higher specific gravity of the richer portion of the ·01 product cause it to pass to the tubes, and the lower specific gravity of the lighter particles of barren rock though ·01, pass on, and thus allow us to obtain the highest scientific and commercial result, while still retaining a comparatively high percentage of ·01? I recognise that this standard will vary on different ores, but we would have a starting point nearer than the complete elimination of the ·01 product.

Mr. Kennedy gives the results of some very interesting experiments, but I think he fails to prove that there is nothing in the cycle theory. He ascribes the result of the difference in favour of the 5 stamps against the 4 stamps, to increased agitation in the boxes, and adds that in the case of the 4 stamps one end of the box was filled in to avoid waste space. I fail to see where the increased agitation came in.

He wishes to know what is to prevent us getting a steady load on the driving belt with 4 cams set equi-distant. A glance at a blue print of 5 cams on a shaft, will convince him that there would be a longer distance between the points of the cams, and consequently a jerk on the belt. He also asks, why not six cams? I think the enormous reduction of the breaking strain of the cam shaft will dispose of that.

He further says that I repeatedly contradict myself in saying that No. 3 should be the longest drop in the box, and Nos. 2 and 4 longer than Nos. 1 and 2, while saying that all stamps should have the longest possible drop. A careful reading of my paper will show him that what I said was that on some orders of drop, at present, Nos. 2, 3, 4, must be longer, and suggested heavier centre shoes to compensate feeder driving and distribution of ore, and thus running all the stamps at the longest possible drop.

Again, "no man on the Rand to-day, with the slightest pretence of being a millman, is guilty of the practice of shortening stamps by inches." Alas! all men are human. He refers me to Gilpin County, Colorado, if I really favour a longer drop. It is unnecessary. Whilst holidaying in Cornwall, I saw a stamp crushing tin ore with a longer drop and running slower than the one he mentions, but I would not dream of recommending that to our Society for good work. What I meant is, the experts on mill construction have given us a cam of a certain length, and it is worth going back a little and trying if it will not do better work when used to its full extent than when it is speeded up and less of it used. If not, get a shorter cam and save money and metal.

Mr. Kennedy also misunderstands my remark on the stamps doing too much work when run in conjunction with tube mills. If the stamps are doing too much work the plant is doing too little, and the management have good cause to be dissatisfied.

In conclusion, I wish to tender my acknowledgments to Mr. G. Andreoli, reduction works officer, Nigel Deep G. M. Co. Though writing in the personal sense, the results given are his.

NOTES ON RAND MINING.

(Read at March Meeting, 1908.)

By Tom Johnson (Member).

DISCUSSION.

Prof. J. Yates: There is one point upon which, with Mr. Johnson's permission, I would like to say a few words, as it is just possible that he may, in replying, overlook it. Mr. Phillips in his contribution says: "But I prefer underhand stoping for machines as then you have better footing for rigging up, and all the holes are water holes, which one cannot have either in overhand or breast stoping." That, of course, is not correct. As a matter of fact, on the Rand mines practically all our holes are wet holes. In fact, in drawing up the rules and regulations in connection with the coming stope drill competitions we stipulated, for the information of competitors, that probably 90% of the holes on the Rand in our stopes are wet holes, notwithstanding the fact that most of our stoping is overhand and breast stoping. (Prof. Yates here gave illustrations of Rand stoping on the blackboard showing that no matter whether it was overhand, breast or underhand stoping the holes were wet holes.) I think the Stope Drill Committee are well within the mark when they say that 90% of the holes are wet holes. It may be that Mr. Phillips has confused our stoping with the practice frequently seen in the United States. There large bodies of ore are often shot down with dry holes drilled above the horizontal. That, of course, is quite a different thing.

Mr. M. H. Coombe (*Member*): In many cases on the Rand to-day the men are overhand stoping in the old-fashioned style, and they are drilling all dry holes. This is absolutely ridiculous when we take into account the average dip of the reefs, and is as uneconomical as it is deleterious to health. This is one grievance the men brought forward in their evidence before the Mining Commission. No mine foreman or shift boss who was a miner himself would ever put a man to drill a dry hole where a wet one could be drilled.

Prof. J. Yates: I have never come across a case such as that mentioned by Mr. Coombe, and I am sure that I am quite correct in saying that it is very unusual. I agree with Mr. Coombe when he states that there is little excuse for adopting that system. It is far better for the health of the men that we should keep to the methods I have sketched on the board.

THE ADAIR-USHER PROCESS.

(Read at May Meeting, 1908.)

By Alfred Adair (Member.)

DISCUSSION.

Mr. H. A. White (*Member of Council*): The thanks of this Society are due to the author for his response to our invitation to give a paper on what is known as the "Usher-Adair Process." Not much interest can be expected to be aroused by that part of the process which involves the use of umber as an oxidising agent, for several reasons. In the first place, all known oxidising agents have been covered by patents already in existence, and a great many have been tried. Only two survive in this country, namely, permanganate of potash and atmospheric oxygen. The latter is the cheaper, and on a working scale is found in practice to be, in the great majority of cases, sufficient. With low grade ores the amount of oxygen readily absorbed by our working solutions is greatly in excess of the theoretical requirements, and in only exceptional cases can it be profitable to add artificial oxygen compounds with their attendant costs and other drawbacks. The authors own experiments on the small scale, especially with current slimes, only really warrant the conclusion, which some large scale results confirm, that no practical difference in extraction can be anticipated. I find that nearly half of the undissolved gold in our slimes residues is not accessible even to aqua regia, which breaks up pyrites and liberates any gold concealed in that manner.

The explanation of the effect of umber offered on p. 335 halts before it reaches its object, and seems to have been lured off the track by the interest awakened in compounds containing manganese and cyanogen.

The Usher washing apparatus, in its modified form as now applied, suffers from some drawbacks, but I would not hastily condemn a process whose theory is so admirable, and which is possibly susceptible to improvements, which may realise in practice many of its founders' dreams. I have had the opportunity of experimenting with it on a working scale, but not with all the present improvements. I see in it the following defects which have not so far been obviated in any plants I have seen at work. The first serious objection is hinted at by the author, and that is the question of temperature, or rather of relative specific gravity of replacing and replaced solution. If the entering solution is even slightly warmer or lighter than that in the treatment tank, it apparently tends to rise in streams through the tank to the overflow and effects very little replacement. This effect I have frequently observed in practice, and I have never seen the efficiency of replacement anywhere near approaching the theoretical. A study of solution values taken at hourly intervals from the overflow will show up this point very clearly to anyone in doubt.

Another drawback is the fact I observed with current slimes, that the solution of the gold in a reasonable time and to, say, the figure of ·1 dwt. undissolved, requires the pulp to be diluted to, say, 3 to 1 of slimes, and this knocks off some of the capacity effect claimed. At the time of my experiments it was claimed that no circulation treatment was required to dissolve the gold, but my results showed conclusively that this was not the case.

Another real objection is that unless the number of pipes placed round the tank is very great, a certain portion of the slime is not treated or washed at all, and this renders the residue sample uncertain. The stratification of the slimes caused by settlement in a slowly uprising current is a very serious drawback to the usual method of slime residue sampling, and will give results much too low if especial attention be not given to this point. After allowing for all these factors, I found a great lack of certainty and regularity in the residues compared with the steady and reliable figures, which is the usual experience with the ordinary decantation method.

With regard to Mr. Adair's all-sliming proposition I would like to point to one objection which the advocates of similar methods have overlooked. That is the fact generally observed, that while the greater part of the gold in our sands plants is readily, practically at once, dissolved, yet some small portion is going into solution even at the end of a six days' treatment, and that, small as this part may be, it is enough to pay expenses of keeping the sand where it is all that time, and even (see vol. vii., p. 411) to pay for transfer to second treatment vats with a sand showing only 7% of ·01 in. grade. If the fact that re-ground sand is not the same thing as ordinary slimes be borne in mind, it will be seen that this factor is not without considerable bearing on the case.

I much regret that the notes on slimes residue assays entirely overlook the importance of the presence of dissolved gold. In this connection my experience has been that the addition of a small quantity of cuprous chloride solution to the sample before drying down will ensure a correct result and differences as high as $\frac{1}{10}$ dwt. may sometimes be observed in comparative fusion with and without the Cu_2Cl_2, though it is possible to get correct results with sufficiently high temperature, the elimination of time-honoured borax, and the securing of a decent sized lead button, without the exercise of this simple precaution, based on Prof. de Wilde's discoveries and Mr. Whitby's adaptations.

Mr. C. E. Rusden (*Member*): On the Geldenhuis Estate we were the first to adopt the improved arrangement of the Usher process. On the assayers returns I can show a 92% extraction on 1½ dwt. slimes, and it also gives us 33% increased tank capacity. I did not use umber in the treatment. I hope to give further details as to results obtained at the next meeting.

Mr. K. L. Graham (*Member of Council*): On reading through Mr. Adair's paper, particularly in that portion dealing with "all sliming," several statements made by him appeared to me so misleading as to demand refutation. Mr. Pearce in his critique, published in the July *Journal*, has, however, dealt with the majority of these, and, to a great extent, has cut the ground from under my feet. As he points out, the reasoning whereby the author arrives at the conclusion that the whole of the battery pulp could be reduced to − 90 for 7½d. to 1s. per ton milled, is based on entirely wrong premises. Moreover − 90 is a very long way off "all sliming," sands from our mills reduced to this degree of comminution give an extraction of from 87% to 90% as the result of 7 to 8 days' treatment. In the scheme outlined by Mr. Adair he claims to treat 150 to 200 tons of fine sands and slimes for 24 hours in three 30 ft. tanks. That he would get an extraction which would correspond at all with that obtained in any of our plants to-day is improbable for the reasons stated in the criticism already referred to. The description of the proposed plant is so extremely visionary as to make it difficult to follow, or to suppose it possible of practical application. It appears to be first cousin to the various continuous slimes processes which have so far proved unsuccessful on these fields, the only novel feature it can claim, as far I can see, being its capacity for treating slimes laden with 75% sands, which by the way, appears to me to be a cumbersome way of describing the pulp from our mills.

I must confess to a feeling of disappointment on reading the paper; notwithstanding Mr. Adair's plucky attempt to make his case for umber good, it has evidently proved not sufficiently beneficial in its effects to make its use general. In that portion written by Mr. Jamieson, the question of gold in solution in slimes residue has been completely ignored, although it is of vital importance when dealing with values as low as ·2 dwt. and under. Practical results from the working of the Usher process on a large scale Mr. Adair has been unable to give us, in view of the conflicting opinions in existence as to the results obtainable, this is particular unfortunate. Personally, I am convinced that where an increase in capacity of plant is desired at a minimum cost, the Usher process will fill the bill, a gain of 30% being calculated on at the Geldenhuis Deep plant, where it is now in course of installation.

Mr. G. Melvill (*Member*): I have a few notes on the practical working of the Adair-Usher process which I will send for publication. With regard to the use of umber I would like to ask Mr. Adair whether he found it had any effect on the alkalinity. I used umber for some time at the Crown Reef mine, and when it was stopped my lime consumption was considerably reduced. This led me to make several experiments with umber on my working solutions as well as the battery pulp, and I proved conclusively that the alkalinity was considerably reduced. This was pointed out to Mr. Durham, the Adair-Usher company's representative, who made similar tests with the same results.

The President: This is rather an interesting point raised by Mr. Melvill, and it will be brought to the notice of Mr. Adair for an expression of his opinion.

Mr. A. Adair (*Member*) contributed: Replying to Mr. Melvill's remarks regarding the supposed decrease in the alkalinity of solutions through the use of umber, it appears to me that, to a trained chemist, an analysis of umber should be a sufficient reply. Transvaal umber contains free carbonate of lime, and has remained in the damp vaults for ages untold. Yet Mr. Melvill believes it to be acid, now. At the Crown Reef $\frac{1}{4}$ to $\frac{1}{2}$% of it, mixed with slimes, was saturated with large volumes of lime water, and excess of lime in the solid form; even that did not cure it. Mr. Melvill makes out that after transfer with a cyanide solution (was it neutral?) the solution had lost alkali, and blames the umber for that loss. The peculiar character of this acid is so chemically new, that I think Mr. Melvill may justly claim the discoverer's right of naming this singular acid. I venture, however, to predict that its ultimate composition will be found to be C. $27{\cdot}\dot{2}\dot{7}\%$ O. $72{\cdot}\dot{7}\dot{2}\%$—a gas with acid properties—known to exist in the atmosphere along with neon, argon, and others.

This explanation is suggested by the method, adopted by Messrs. Melvill and Durham, in proving the acidity of umber: they determined the excess of caustic alkali in the dilute solution, by allowing it to remain overnight in an open beaker, subsequently titrating with an unstandardized standard solution.

Not a single word on this subject passed between Mr. Melvill and myself, nor was I aware of his anxiety "to prove it to me," but it is quite true, that I never *went* for his proof, at least before.

BATTERY AND CYANIDE GOLD SMELTING.

(Read at July Meeting, 1908.)

By A. Thomas (Member).

DISCUSSION.

Mr. H. A. White (*Member of Council*): I congratulate the author on his very useful paper, and am sure that there is room for several more similar contributions to our proceedings. If standard practice in any branch of our metallurgy is carefully described and presented for our consideration the author's aim must tend more towards detailed accuracy than mere novelty, but I am sure that papers on these lines will not be among the less welcome or useful to our members.

It is, however, on these smaller details that useful discussion may be based and different methods compared. For example, in retorting gold I have found it very convenient to use a tray made in one continuous piece to fit the retort and yet leave a space of about 4 in. clear inside the lid where the heat is usually so low as to retard the operation if amalgam be allowed there. In such a tray 7,000 oz. of amalgam may be accommodated without fear of overflow. Both the tray and the interior of the retort may be washed with lime and the pieces of amalgam are separated by brown paper. The tray will last a very long time, and is made by dishing to shape a sheet of $\frac{1}{4}$ in. mild steel plate. This operation is not exactly easy, but Mr. Stanbury, foreman blacksmith at the May Consolidated, found no insuperable difficulty and would explain the method to anyone interested.

I do not see the necessity for having the bearers water cooled, but the flue arrangement mentioned is imperative in order to heat the retort as evenly as possible. We find that stout asbestos sheet makes a very suitable joint if wetted before use.

The process described for reduction of gold slime seems somewhat illogical, as the slags first produced have afterwards to be re-treated in the pan furnace, and the lead derived therefrom must be cupelled. Incidentally the author shows a very large loss of silver.

By direct "Tavener" process it would seem that the drying and pot smelting would be entirely eliminated, and it is not at first sight obvious that any advantage is gained by the interpolation of these tedious and expensive operations with their attendant risks of loss—estimated by a Past-President of this Society at about 3% of the gold present. The size of the pan furnace required for a large plant has been mentioned as an obstacle, but it would be quite simple to duplicate this part of the apparatus if necessary, though very large reverberatory furnaces are used for other metals without serious drawback in their working.

From a practical point of view, however, the advantage of securing the greater part of the month's output with absolute certainty at a definite date, and preventing large quantities of gold being locked up in the bottom of a furnace, as has frequently been noted in the Tavener process, can readily be realised. In a mine where large reserves of gold are maintained beside that locked up in zinc boxes and on copper plates, my present opinion is that the advantages of the pan furnace, pure and simple, far outweigh any other at present known method of smelting. At the May Consolidated the present pan furnace bottom has lasted nearly three years and still looks good and produces about 1 ton of slag a month of an average value of 6 dwt. gold and much less silver. Any slag over an ounce in value goes into the next smelt.

There is one point involved, though not actually mentioned in Mr. Lee's contribution to this discussion, to which I would like briefly to refer. That is the practice in some mines of requiring the assayer to preside over or at least assist in the smelting of rich products. When I was an assayer I strongly and successfully objected to this practice on the score of risk of contamination of the assay office and other reasons. I still prefer that the cyanide and battery managers should handle their own products right up to the point of marketable bars of bullion, and now-a-days it must be rare to find men in these positions who are not thoroughly capable in such matters. The assayer is only handed samples in such conditions as to cause no risk of contamination, and even such a sample as that of zinc returned to cyanide boxes is treated at the cyanide works and the washed product folded up in filter paper and placed in a pot with flux ready for the furnace.

The President: There is one point that struck me in Mr. Thomas' paper, and that is where he mentions the size of his flues in the retort furnace. Of course, this is a matter that depends on the height of the chimney you have. The author's flues are 11 in. × 8 in. from which, I presume, the length of his smoke stack is very small. I have generally been in the habit of having a smoke stack of about 30 ft., and I found a very much smaller flue was necessary, say, 6 in. × 4½ in. for the front and back flues, and the centre flue was very much reduced, being not more than 2 in. wide. If you have them all of the same dimensions you will have a decided draw towards the centre. I have found by experience that it is advisable to reduce the dimensions of the centre flue very much so as to check the drawing toward the centre. I think, roughly, the dimensions of the flues or openings into the smoke chamber are 6 in. × 4½ in. for the front and back, and 6 in. × 2 in. in the centre. That gives a very even distribution of the heat.

The meeting then closed.

Contributions and Correspondence.

SILVER ORE TREATMENT IN MEXICO.

In Mr. Caldecott's excellent paper on "Silver Ore Treatment in Mexico" he refers to the detrimental effect of manganese, which is occasionally found associated with the silver. The manganiferous silver ores form a distinct class from those ores which are being so successfully cyanided in Mexico. The cyaniding ores are those which contain the silver, either free or as sulphide or chloride, and the extraction of silver from such ore has been brought to a very high rate through modifications of treatment that have been introduced. The extraction from manganiferous ores on the other hand is low, and but little affected by those modifications of treatment that have been attended with good results in treating the non-manganiferous ores.

The presence of manganese has in itself no hindering effect on the dissolving of silver in cyanide solutions, if the silver be present in some soluble form and the manganese as a constituent of the ore that is separate and uncombined with the silver. The rebellious character of the manganiferous silver ores of Mexico is due to the silver being locked up in a manganese compound in such manner as to prevent the action of the cyanide solution upon it. If the manganese compound is first broken up and the silver dissociated from it, there is no difficulty in extracting the silver, the manganese still being present.

To illustrate this I may cite a series of tests I made on silver extraction from two classes of ore taken from the same mine. One contained 2% to 6% Mn, and carried 12 oz. to 60 oz. silver; the other, no manganese and up to 4 oz. silver.

The non-manganiferous ore cyanided readily, the extraction averaging 68% on several tests with ·2% KCy solution without any modification of treatment. Another example of the same ore, when mixed with 15% MnO_2 and treated with the same cyanide solution, gave an extraction of 69%, showing that the presence of uncombined manganese exercised no effect on the solvent power of the solution.

The manganiferous ore on the other hand proved very rebellious, representative tests being as follows:—

			%			%
20 oz.	ore	treated with	·075	KCy sol.	Ag. extr.	17
20	,,	,,	·5	,,	,,	18
20	,,	,,	1·0	,,	,,	16
60	,,	,,	·075	,,	,,	13
60	,,	,,	·5	,,	,,	11
60	,,	,,	1·0	,,	,,	10

The addition of lead acetate, litharge, mercuric chloride or mixed bromides to the treatment failed to effect any higher extraction than was effected by the unmodified cyanide treatment. All tests were made on ore ground to pass 150 mesh, and treated with thorough agitation and aeration until 24 hours treatment failed to show any further extraction. Notwithstanding that all of these tests showed so low an extraction, when the ore was given a preliminary treatment to break up the manganese compound, and this followed by cyaniding with ·2% solution, gave uniformly good results, the extraction of silver in several tests being over 90%.

ROBERT LINTON.

Magdalena, Sonora, Mexico,
July 1, 1908.

[Mr. Linton has been asked to kindly furnish a description of the "preliminary" treatment to which he refers.—ED. COM.]

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

SEPARATION OF VANADIUM AND MOLYBDENUM.—"Continuing the work of Gooch and Edgar, the author finds that whilst, as is well known, vanadic acid is readily reduced by sulphur dioxide to the condition of tetroxide, molybdic acid is not affected by sulphur dioxide even at a concentration of 0·4 gm. of molybdenum trioxide in 25 c.c., provided there be present at the some time 5 c.c. of concentrated sulphuric acid. To determine the two acids together, then, a quantity containing about 0·4 gm. of the mixed acids is diluted to 75 c.c., 2—3 c.c. of concentrated sulphuric acid are added, the liquid is heated to boiling, and a current of sulphur dioxide is passed through till the reduction of the vanadium compound is complete. Carbon dioxide is now passed through, and the liquid boiled till the sulphur dioxide is completely expelled; the reduced vanadium tetroxide is now titrated with *N*/10 potassium permanganate, which converts it into the pentoxide. The solution is then passed through a Jones' reductor (preceded by 100 c.c. of hot water and 125 c.c. of 2½% sulphuric acid, and followed by 100 c.c. of the sulphuric acid and 200 c.c. of hot water), and received in iron alum solution (50—60 c.c. of 10% solution), which, after addition of 8—10 c.c. of syrupy phosphoric acid to decolorise the ferric salt, is titrated with *N*/10 permanganate. The amount of permanganate required for the vanadium (V_2O_2 to V_2O_5) is three times that used in the former titration (V_2O_4 to V_2O_5), and the remainder is that required for the molybdenum (Mo_2O_3 to MoO_3)."—G. EDGAR, *American Journal of Science*, 1908, *25*, 332—334.—*Journal of the Society of Chemical Industry*, May 15, 1908, p. 472. (A. W.)

TITANOUS SULPHATE AS A REDUCER OF IRON IN VOLUMETRIC ANALYSIS.—"The iron compound is dissolved in sulphuric acid; to the solution in the cold is added titanous sulphate solution in slight excess, bismuth oxide added to destroy excess of titanous sulphate, the solution filtered into about a litre of cold distilled water, and titrated with *N*/10 permanganate. The author prepares the titanous sulphate by fusing 20 gm. of commercial titanic acid with potassio-sodium carbonate, dissolving the ground reaction-product in hot concentrated sulphuric acid, filtering, reducing by excess of zinc, and filtering into 2 litres of water. The solution is best kept in an atmosphere of hydrogen, as suggested by Knecht."—H. D. NEWTON, *American Journal of Science*, 1908, *25*, 343—345.—*Journal of the Society of Chemical Industry*, May 15, 1908, p. 472. (A. W.)

DETERMINATION OF VANADIUM.—"The ore treated at the mill of the Vanadium Alloys Company, at Newmire, Colo., is a vanadiferous sandstone containing usually a very small per cent. of uranium and iron and only a trace of arsenic. When appreciable quantities of arsenic or molybdenum are encountered, these elements are separated in sulphuric acid solution by means of hydrogen sulphide. For the ordinary determination of vanadium the method employed at the laboratory is as follows:—

Weigh 1 gm. of the finely pulverised ore into a No. 3 casserole; add 40 c.c. dilute nitric acid (one to one), and evaporate on the water-bath. Vanadium is indicated by the formation of dark red rings around the sides of the dish. Evaporate to complete dryness, cool, and add 20 c.c. dilute nitric acid and 3 c.c. concentrated sulphuric acid and evaporate until copious fumes are evolved; cool and take up with 30 c.c. cold water. Boil in the casserole until all vanadium is in solution, stirring constantly to avoid loss by bumping. Some vanadium is not dissolved by the strong sulphuric acid, but readily goes into solution upon boiling with the dilute acid. Filter into a tall beaker and wash the silica well with hot water. Boil the filtrate down to a bulk of about 30 c.c., cover with a watch glass, and add hydrogen peroxide carefully to the hot solution until a permanent deep red colour is obtained. Let the mixture stand a moment and then add a concentrated solution of caustic soda until the precipitation of the iron is complete. Dilute to about 200 c.c. with boiling water, and boil briskly for five minutes. Let the mixture stand hot until the precipitated ferric hydroxide has settled, and filter rapidly through a fluted funnel, allowing the filtrate to run into a 600 c.c. Erlenmeyer flask. Wash six times with hot water, dilute to about 350 c.c., and acidify with concentrated sulphuric acid. Heat the solution to boiling and reduce with strong, freshly prepared sulphur dioxide water; enough should be added to give a decided odour of sulphur dioxide and a distinctly blue colour to the solution. Boil about 20 minutes to drive off the excess of sulphur dioxide, passing carbon dioxide gas through the solution continually to aid in the expulsion of sulphur dioxide and avoid any possibility of reoxidation of vanadium to the pentoxide form. Titrate hot with potassium permanganate, the standard of which is approximately 1 c.c., equivalent to 0·005 gm. vanadium.

Uranium does not interfere in this method, being precipitated along with iron. The separation of the vanadium from these two metals by means of caustic soda is practically complete in one precipitation, whereas in the usual method employing ammonia, two or more precipitations are necessary to effect a complete separation.

For a check method the following is sometimes used:—Fuse 1 gm. ore with 3 to 4 gm. sodium carbonate in a large platinum crucible, keeping the crucible covered, until a quiet melt is obtained; then remove the cover and heat five minutes at a bright red heat to insure the conversion of all the vanadium to the pentoxide. Cool and add about ¼ gm. finely powdered sodium nitrate and heat again carefully to a red heat (never hotter) until the fusion is complete. Remove the cover of the crucible and heat a few minutes longer. Cool and leach the melt in a beaker with hot water, boiling, if necessary, to break up all the lumps, and filter into an evaporating dish. Wash the precipitate four times with hot water, best by boiling in the beaker and decanting. Finally, cool the filtrate and add carefully concentrated sulphuric acid until acid. Evaporate until the silica settles out as a slimy precipitate; filter and wash 10 or 12 times by boiling the precipitate in the evaporating dish and decanting upon filter paper. Allow the filtrate and washes to run into a 600 c.c. Erlenmeyer flask; dilute the final solution to about 350 c.c. with hot water, reduce and titrate with permanganate as described. Uranium does not interfere.

In case very exact determinations are required blanks are usually run. A quantity of acid and sulphur-dioxide water equal to that used for the regular sample is added to the blank and the two assays are boiled and titrated simultaneously. The amount of permanganate required for the blank is then substracted from the burette reading on the regular sample. Errors due to impurities in the solution of sulphur dioxide are thus eliminated."—A. M. Wilson.—*Engineering and Mining Journal*, May 9, 1908, p. 962. (G. H. S.)

Ammonium Phosphomolybdate in the Determination of Phosphorus in Iron and Steel.—"Quantities of sodium phosphate containing from 6·648 to 0·277 mgm. of phosphorus, were precipitated by 50 c.c. of a reagent containing in a litre 50 gm. of ammonium molybdate and 500 c.c. of nitric acid sp. gr. 1·20, and similar quantities by 50 c.c. of the same reagent together with 5 gm. of ammonium nitrate. The liquids were kept at 45° C. for 2½ hours. In the first series, the proportion of phosphorus in the precipitate increased, as the amount of phosphate was lessened, from 1·69 to 2·25%, whilst in the second series it decreased from 1·62 to 1·28%. Similar differences in composition occurred during precipitation in presence of ferric salts, and were intensified by the co-precipitation of ferric molybdate when ammonium nitrate was present. In these cases it is desirable to dissolve the precipitate in ammonia and re-precipitate by nitric acid. Besides this precaution, the following details should be attended to in determining phosphorus in iron or steel: for each gram of metal 50 c.c. of molybdate reagent and 5 gm. of ammonium nitrate should be used; precipitation should take place at 65°–70° C. (1½ hours); the precipitate should be redissolved in the same quantity of ammonia (50 c.c. of 1 : 4), re-precipitated by concentrated nitric acid, and after complete precipitation, 15 c.c. of the molybdate reagent should be added; the liquid and precipitate should be kept for 2 hours at 40° C., and after filtration, the precipitate should be washed with pure water. The precipitate so formed, after drying at 105° C., contains 1·60% of phosphorus"—G. Chesneau, *Compt. rend.*, 1908, *146*, 758–761.—*Journal of the Society of Chemical Industry*, May 15, 1908, p. 473. (A. W.)

Electrolytic Separation of Nickel and Copper.—"Nickel can be accurately determined by electrolysis of the nitrate, provided (1), no nitrite be present, (2), a sufficient excess of ammonia be added, and (3), a straight wire of passive iron be used as anode (platinum anodes are more or less irregularly attacked, and the dissolved platinum is deposited with the nickel). Accurate separation of nickel from copper, and determination of both metals, can thus be effected. The metals (0·4–0·5 gm. in all), or compounds containing them, are dissolved in nitric acid, so that the solution contains about 5 c.c. of the strong acid in excess; any nitrous acid is expelled by boiling, or destroyed by the addition of urea in not too great excess, the volume made up to 100 c.c., and the copper precipitated by a current of $ND_{100} = 1$ ampère at the ordinary temperature, warming towards the end of the operation (1½—2 hours) not above 50°–60° C. To the bath is now added 80 c.c. of ammonia (sp. gr. 0·91), the iron wire anode (cleaned in hydrochloric acid and rendered passive in strong nitric acid) substituted for the platinum one, the coppered cathode replaced, and the nickel deposited on it by a current of $ND_{100} = 5$ ampères; the current heats the liquid to about 70° C. When the liquid is decolorised and gas evolution occurs at the cathode, the latter is withdrawn without stoppage of current, washed, dried and weighed. The anode is again rendered passive, and the electrolysis continued for another 5 minutes, when the cathode should not have increased in weight. The deposition of the nickel should not need more than 50–55 minutes. The ammonium sulphide test for complete removal of the nickel is useless in solutions containing so much free ammonia, hence the second electrolysis and weighing."—A. Thiel, *Z. Elektrochem.*, 1908, *14*, 201–208.—*Journal of the Society of Chemical Industry*, May 15, 1908, p. 473. (A. W.)

Stable Calcium Hypochlorite.—"The commercial importance of bleaching powder depends on the fact that it is a convenient portable form of chlorine, and that by suitable treatment with an acid it may be caused to yield this element in a highly concentrated and active form. It has, however, the disadvantage of being rapidly decomposed by atmospheric moisture, and slowly loses its strength, even when preserved in stoppered vessels. Moreover, since the absorption of chlorine by slaked lime is not complete it never evolves more than 38% of this element on treatment with an acid, and the proportion of chlorine set free is usually considerably less. Only about half the chlorine in the bleaching powder is 'available,' the rest being chiefly present in the form of chloride. A process has recently been described for preparing in a state of purity calcium hypochlorite, the active constituent of bleaching powder. Milk of lime is saturated with chlorine, and the solution thus obtained evaporated as rapidly as possible in a vacuum. Calcium hypochlorite is precipitated in the concentrated solution, and thus separated from calcium chloride, the other product of chlorination. The precipitate is unstable owing to the presence of adhering solution and water of crystallisation. Accordingly it is dried in a vacuum until the whole of the moisture and a portion of the chemically combined water are eliminated. The final product, which is quite stable, does not deliquesce in moist air; it dissolves in water to a clear solution, and when treated with hydrochloric acid evolves 80 to 90% of its weight of chlorine. The production of this salt at a fairly reasonable price would have a far-reaching effect on the technical applications of chlorine. To obtain a given result in chlorine bleaching or any other similar operation involving the use of chlorine only half as much of the new salt would be required as of ordinary bleaching powder. (German patent, 188,524.)"—*Times Engineering Supplement*, April 29, 1908. (J. A. W.)

OXIDISABILITY OF PLATINUM. — "The employment of platinum in the construction of apparatus liable to be brought into contact with corrosive chemicals is based on the inalterability of the metal towards powerful reagents. Nevertheless, the metal does under certain conditions undergo a certain amount of corrosion, and the actions of fused caustic potash and concentrated sulphuric acid on platinum vessels have already been referred to in these notes (*Engineering Supplement*, July 18, and October 24, 1906). In two recent numbers of the *Comptes Rendus de l'Académie des Sciences*, 1908. pp. 117 and 475, C. Marie draws attention to the oxidisability of platinum in aqueous solutions at the ordinary temperature. When left in contact with alkaline permanganate or ferricyanide, platinum strips become coated to a slight extent with a thin layer of oxide, which, when dissolved subsequently in hydrochloric acid gives a characteristic coloration with potassium iodide. Permanganate also oxidises the metal in acid solution, and so do potassium persulphate, bichromate, and chlorate. Hot concentrated nitric acid in particular has a marked action on platinum, particularly when warmed. Acid solutions of ferric chloride or hydrogen peroxide, on the other hand, have no action on the metal, and the peroxide is also inert in alkaline solution. Although the amount of platinum oxide produced in this way by direct oxidation of the metal is only small, yet the possibilities of contamination by traces of platinum salts should be borne in mind in exact experiments. An alloy of platinum and iridium containing 20% of the latter metal is also affected by oxidising agents."—*Times Engineering Supplement*, April 29, 1908. (J. A. W.)

THE PRODUCTION OF HYDRAZINE FROM INORGANIC SOURCES.—"Hydrazine is a powerful nitrogenous base having the formula N_2H_4, which was first isolated by Curtius from a complex organic compound called ethyl diazoacetate. It has since been prepared from purely inorganic materials. By the interaction of potassium bisulphite and nitrite a complex dinitrososulphonate is obtained, which, when successively reduced with sodium amalgam and distilled with caustic potash, furnishes hydrazine. More recently Raschig has discovered a very simple process for preparing the base from sodium hypochlorite and ammonia, which will probably lead to the production of the compound on a commercial scale. When ammonia and a hypochlorite interact, chloramine NH_2Cl is produced. On warming the solution of this substance with ammonia one or both of two changes may occur, the two reagents mutually decompose each other, forming nitrogen and hydrochloric acid, or they combine to yield hydrazine hydrochloride N_2H_4, HCl. The latter reaction is promoted and the former inhibited when a viscous substance such as joiners' glue is introduced into the solution. In these circumstances a yield of 40 to 50% of hydrazine can be obtained and isolated in the form of its sparingly soluble sulphate. Providing that this base can be produced at a moderate price there can be little doubt but what it will find considerable employment in the manufacture of fine chemicals as well as in analytical chemistry. The above reaction is the most promising from the economical standpoint owing to the cheapness of the initial materials."—*Times Engineering Supplement*, April 29, 1908. (J. A. W.)

AUTOMATIC ESTIMATION OF CO_2 IN FURNACE GASES.—"Dr. C. A. Keane and Dr. H. Burrows described their experiments with the autolysator, an apparatus for the automatic estimation of carbon dioxide, devised by Strache, Johoda, and Cenzkin, which differs from all other forms of automatic analysers in giving a continuous, instead of an intermittent, record of the carbon dioxide content of the furnace and other gas examined. The gases were drawn continuously through reservoirs containing soda-lime and the diminution in pressure caused by the absorption of the carbon dioxide was recorded at any instant by specially constructed differential manometers. The results showed that the records of the instruments agree within 0·3% with those obtained by the ordinary analytical methods. Moreover, the instruments respond very efficiently to the changes in the percentage of proportion of carbon dioxide. Experiments were also conducted on the furnace gases from boilers and the conditions of working examined, more especially in regard to the methods of withdrawing the sample and to the life of the absorbent employed."—*Times Engineering Supplement*, June, 1908. (J. A. W.)

VOLUMETRIC ESTIMATION OF COPPER AS CUPROUS THIOCYANATE BY POTASSIUM IODATE.—"The titration of copper as cuprous thiocyanate with potassium iodate is recommended as an easy, rapid, and accurate method for the estimation of copper. The ore is decomposed and the copper converted into sulphate in the usual manner, silver, if present, being precipitated with a little hydrochloric acid before filtering. The filtrate is neutralised with ammonia, 10—15 c.c. of strong sulphurous acid solution added, and the whole boiled. 5—10 c.c. of a 10% solution of ammonium thiocyanate are added, according to the amount of copper present, and after stirring, the precipitate is allowed to settle and is then filtered off and washed with hot water. The filter with its contents is placed in a 250 c.c. glass stoppered bottle with 5 c.c. of chloroform, 20 c.c. of water and 30 c.c. of concentrated hydrochloric acid, and titrated with a standard potassium iodate solution, agitating well between additions.

The violet colour imparted to the chloroform disappears at the finish with great sharpness. Solutions of potassium iodate keep indefinitely, and the pure salt may be readily procured. Each cubic centimetre of a solution containing 5·892 gm. per litre, is equivalent to 1 mgm. of copper."—G. S. JAMIESON, L. H. LEVY and H. L. WELLS, *Journal of the American Chemical Society*, 1908, *30*, 760–764.—*Journal of the Society of Chemical Industry*, June 15, 1908, p. 572. (A. W.)

EXACT RELATIONS AMONG THE ATOMIC WEIGHTS. In an interesting paper by D. F. Comstock on the indestructibility of matter and the absence of direct relations amongst the atomic weights, a quotation and recalculation on the basis of the International Atomic Weights from a paper by Rydberg (*Zeit. Anorg. Chem.*, 14, 66, 1897), is given in which it is shown that the atomic weights of the first twenty-seven elements of the periodic system approximate to whole numbers very much more closely than chance could bring about. Also that the atomic weights of these elements are best considered as the sum of two parts (N + D), where N is an integer and D is a fraction, in general positive and smaller than unity. If M is the number of the element in the system, then N = 2M for elements of even valency, and 2M + 1 for elements of odd valency. Below is a table showing this:—

Symbol.	M.	2M.	2M + 1.	Atomic Weight.	D.
He	2	4		4	
Li	3		7	7·03	·03
Be	4	8		9·1	1·1
B	5		11	11·0	—
C	6	12		12·00	—
N	7		15	14·01	–0·99
O	8	16		16·00	—
F	9		19	19·0	—
Ne	10	20		20·0	—
Na	11		23	23·05	0·05
Mg	12	24		24·36	0·36
Al	13		27	27·1	0·1
Si	14	28		28·4	0·4
P	15		31	31·0	—
S	16	32		32·06	0·06
Cl	17		35	35·45	0·45
A	18	36		39·9	3·9
K	19		39	39·15	0·15
Ca	20	40		40·1	0·1
Sc	21		43	44·1	1·1
—	22	44		—	—
—	23		47	—	—
Ti	24	48		48·1	0·1
V	25		51	51·2	0·2
Cr	26	52		52·1	0·1
Mn	27		55	55·0	—
Fe	28	56		55·9	–0·1

The orderly arrangement of this series is very striking, and only in three cases is D greater than unity and only in two cases negative. The heavier elements do not conform well to this scheme but, as Rydberg points out, this is not a valid objection for the numerical values of the weights of the heavier elements depend much more on the value of the arbitrary unit chosen than do those of the lighter weight elements, and hence they can have little influence one way or the other in estimating the validity of the curious relations here set forth. The whole question is, are these merely mathematical remainders or do they represent real physical deviations?—*Journal of the American Chemical Society*, xxx., v., 683, May, 1908. (J. A. W.)

METALLURGY.

HISTORICAL SUMMARY OF THE CYANIDE PROCESS.

Diesbach ...	1704	Discovers Prussian blue.
Macquer ...	1752	Discovers ferrocyanide of potassium
Scheele ...	1782	Discovers prussic acid.
Hagen ...	1806	Predicts that potassic cyanide will dissolve gold.
Gay Lussac	1815	Isolates the radical cyanogen.
Elkington	1840	Discovers that KCN plus electricity dissolves gold.
Bagration	1843	Discovers that KCN alone will dissolve gold.
Elsner ...	1844	Potassic cyanide a gold solvent.
Faraday ...	1857	That gold leaf dissolves much faster in cyanide solutions when exposed to the action of air.
Elsner ...	1866	That oxygen is necessary to dissolve gold, and gives the now well-known Elsner formula.
Wurtz ...	1866	Discovers that dilute cyanide solutions dissolve gold.
Rae ...	1867	Patents a cyanide process for dissolving gold in ores with the aid of electricity.
Simpson ...	1885	Patents a cyanide process for the same purpose, using alkali.
McArthur-Forrest	1887	The cyanide process patented, basic claim dilute solutions.
	1889	The first cyanide mill, erected in New Zealand.
	1890	The first tailing plant, erected on the Rand, Transvaal.
Paul ...	1891	Wet crushing with cyanide solution in batteries.
Gaze ...	1892	Discovers the benefit of using bromo cyanogen in solutions.
Argall ...	1893	Introduces and patents the separation of dust and slime from sands, subjecting each to a separate treatment.
MacArthur	1893	Patents the use of lead salts in KCN solutions.
Argall ...	1895	Introduces the roasting of sulphotelluride ores as a preparation for cyanide treatment.
Williams...	1896	Introduces the decantation process for slime treatment.
Sutherland	1898	Introduces the filter press process of slime treatment and also the tube mill. Dr. Diehl claims to have used these mills in 1896 on Kalgoorlie ores in Hamburg.
Moore ...	1903	Introduces the movable vacuum filter with fixed tank.
Cassel ...	1904	Introduces the fixed vacuum filter in fixed tanks.
Ridgway ...	1905	Revolving automatic suction filter.
Argall ...	1906	Introduces the gravity suction filter with fixed filters and movable tank

—PHILIP ARGALL.—*Mines and Minerals*, March, 1908, p. 368. (W. A. C.)

ALLOYS OF GOLD AND TELLURIUM. — "The mixtures of Au and Te were made by melting Te or a Te-Au alloy in a clay crucible under charcoal and adding a weighed quantity of Au. Some gravitic separation took place; and the lowest part of the culot contained about 1% more Au than the highest. Methods of assaying the mixtures are described. The solidification curve shows a maximum at 452°, corresponding to the compound $AuTe_2$, which consists of gold 43·59% and tellurium 56·41%.

The main conclusions are: (1) A compound,

$Au\,Te_2$ or $Au_2\,Te_4$ corresponding in composition to sylvanite or calaverite is formed when Au and Te are melted together in certain proportions, M.P., 452°. (2) Two eutectic mixtures are formed corresponding to Au Te and $Au\,Te_6$; under the microscope they do not show the characteristic banded eutectic structure, but there are no certain indications that they are true compounds. (3) All alloys of Au and Te are brittle. (4) All those containing less than 60% of gold melt between 397° and 452°."—T. K. Rose, *Inst. Mining and Metallurgy.—Electrochemical and Metallurgical Journal*, April, 1908, p. 156. (A. McA. J.)

Separation of Gold from Auriferous Platinum.—"To separate gold from auriferous platinum or other similar substances, Mr. E. Schardin, a Russian chemist, divides the substances in question into several lots, according to dimensions of the grains. Each of them is then separately treated as follows. The material is made into a stiff paste with water, and then mixed with a requisite amount of mercury. To separate the platinum from the gold amalgam and free mercury, all is washed in a current of water. The amalgam is then placed on a flat vessel and stirred with a metallic brush, to which the gold amalgam adheres. The platinum remains unmixed."—*Anales de Chimie Analytique.—London Mining Journal*, May 2, 1908, p. 534. (A. R.)

New Use for Plaster of Paris.—"Not only is plaster of Paris an excellent flux for melting such materials as washings, grindings, or chips, but it has also the property of removing iron, a fact not generally appreciated. Plaster of Paris when melted fuses to a clear liquid, which has the property of dissolving oxides. Being a neutral flux it does not attack the crucible. To use plaster of Paris for melting washings, grindings, or other scrap which contains iron, about 5 lb. are mixed with 100 lb. of the scrap and the whole is melted down. The plaster melts and clears the scrap of the iron. When the whole is thoroughly melted and the metal well stirred the plaster need not be skimmed off. The whole is poured into an ingot mould. When cold, the plaster is easily removed by a blow from a hammer. Frequently it comes away of its own accord. Small quantities of iron may in this way be removed from scrap metals, a good melt being at the same time obtained. Not only is plaster of Paris an excellent flux for melting such materials as washings or grindings, but it has the great advantage of not attacking the crucible. This is not possessed by any other flux."—*Indian Engineering*, March 14, 1908, p. 175. (A. R.)

Settling Slimes in Cyanide Treatment.—"The author describes a process which has given remarkable results, both in the completeness of the separation effected and in the small proportion of liquid carried off by the solid matter, which is removed as it reaches the bottom of the tank. This is of inverted pyramidal shape, and connects at the bottom through an 8-in. square aperture with a closed box in which a 10-in. belt is made to travel slowly. The method may be applied in practice in either of two ways. First, as an intermittent process in which the belt discharge of solids would be delivered into a second similar tank where a weak solution or wash water could be sprayed on to the belt to loosen and disintegrate the slime ; or a mixer may effect this purpose. After removal of the required proportion of solids the solution would be withdrawn to the original level and a second charge introduced. Secondly, the introduction of a suction filter near to the surface of the liquid in the separator, permits of a continuous process being carried on. The percentage of moisture in the discharge is inversely as the proportion of solids in the separator charge, and the first portions of a discharge from any given separator charge are always better than the succeeding portions, hence if the density of the separator charge is maintained at its initial figure, the moisture in the discharge would be kept at a minimum and the rate of discharge increased. By an automatic periodical cutting off of the suction and application of small back pressure the filter screen is kept clear from caked slime and clear liquid is continually drawn from a pulp of sp. gr. 1·4. The belt and suction must be so correlated that the solids discharged are proportionate to the liquid sucked out, and the discharge from a pulp of the density stated will uniformly contain about only 22·5% of moisture. If there be two suction pipes with screens they are kept clear by back pressure being applied alternately to each. The continued use of screen suction would not be possible unless the solids were being extracted by the conveyor belt. The procedure in the wash settlers is practically the same. The solids are washed off the belt by precipitated solution and water, and the suctions withdraw an equal amount of liquid, which is passed through the extractor boxes and returned from the sump. In other processes completeness of washing is dependent on time and capacity of plant, as each instalment of slime is taken separately and washed solution passed through it as long as economically possible, when it must be removed, whether well washed or not, to make room for the next instalment ; but in this method continuity is unimpeded and slimes may be washed out with any reasonable quantity of weak solution without affecting the time of treatment—the only factor affected being the capacity of the extractor boxes.

The advantages of this method are : (1) Absence of pumps, carrying machinery and complicated appliances ; (2) the separation of sands from slime is not called for, and thus disadvantages attendant on dealing with such classes of material as depend on close classification are obviated ; (3) it is possible to wash to the best advantage and thus reduce the loss by residual moisture ; (4) the degree of fineness or uniformity is immaterial ; (5) the initial cost is very low, and niceties of adjustment are not requisite ; (6) the only limit to the thickness of a charge is the point at which it will not run, and best results are obtained from charges with high percentages of solids ; (7) the process may be continuous. The paper is accompanied by diagrams and tables showing the arrangement and the results of working."—H. G. Nichols, *Inst. Mining and Metallurgy.—Electrochemical and Metallurgical Journal*, April, 1908, p. 156. (A. McA. J.)

High Temperature.—"The fact that high temperatures are now daily obtained by means of the electric furnace, the oxy-acetylene blowpipe and other artificial means causes an interest to be manifested in the actual temperature that is reached in melting refractory materials in the sun, whose temperature is the highest known. Experiments with the optical pyrometer recently made are of interest as indicating the temperatures that are reached in certain operations and in the sun. The results are as follows:

Porcelain furnace, 2,498° F. ; glass furnace, 2,552° F. open-hearth steel, 2,795° F. ; melted platinum, 3,236° F. ; incandescent lamp, 3,272° F. ; arc lamp, 7,410° F. ; the sun, 13,712° F."—*American Machinist*. (A. R.)

OXIDATION OF METALS.—"Messrs. Jordis and Rosenhaupt recently studied the question of oxidation of various metals (copper, zinc, tin, and alloys of zinc and tin with copper) by dry or moist air and oxygen. The metals are not attacked at the ordinary temperature. Above 60° C. copper is attacked; above 100° C. tin; above 180° C. zinc. Below 200° C. copper and zinc are attacked a little less rapidly by moist oxygen than dry, and above 200° C. the inverse occurs. It is the same with tin above and below 200° C. Air reacts more energetically than oxygen on copper. This influence of moisture and nitrogen can be explained by formation of small quantities of oxygen peroxide and oxygenated nitrogen compounds which make oxidation more active. Rapidity of oxidation depends also upon the exterior aspect of the metal and physical properties of the oxide formed. The definite alloys act differently to their constituents; but if one of the metals is in excess the authenticated properties of this metal are observed."—*La Nature*, May, 1908.—*London Mining Journal*, June 27, 1908, p. 781. (A. R.)

ECONOMICS OF TUBE MILLING.—"An examination of the literature of tube milling shows that the subject of power consumption has been more open to discussion than any other point in the operation of this class of machines. Different operators, apparently operating under very similar conditions, report their power consumption varying from 35 h.p. to 100 h.p. for the same general type of mill. The manufacturers of tube mills also seem to be at wide variance as to the power that will be required to operate their several makes of mill. The articles that have been published on this subject have neglected to state in detail the different conditions that might have affected the reported power consumption. In view of this lack of detail and the varying results recorded, the details of a series of tests made by the writer at the Colorado plant, Colorado Springs, Colorado, may be of interest.

TABLE I.—Showing per cent. ore and solution, fineness of grinding, and hors-power, with a varying pebble volume while ore and solution feed remained fixed.

Test No.	Pebbles lb.	On 60.	On 100.	On 150.	Thro'gh 150.	Ore. percent	Solution percent	Tons Ore per 24 hours	Indicated Horse-power.
1	3,000	42·5	27·5	8·0	22 0	63·72	36·28	172	18·80
2	6,000	46·5	23 5	8·0	22·0	70·17	29 83	172	20·37
3	9,000	42·0	26 0	8·0	24·0	74·29	25·71	172	22·50
4	12,000	32·0	32·0	12·0	24·0	60·00	40·00	172	32·16
5	15,000	29 0	30·0	14·0	27·0	65·38	34·62	172	39·13
6	16,800	18·0	36 0	12·0	34·0	66·67	33·33	172	42·88
7	18,000	3 5	29 0	16·0	51·5	66·67	33 33	172	47·16
8	19,000	4·0	28·0	13·0	55·0	66·67	33·33	172	51·45*
9	20,000	9·0	32·0	15·0	44 0	71 88	28·12	172	56·28
10	21,000	6 0	30·0	13·5	50·5	71 88	28·12	172	60 10
11	22,000	6 0	29·0	15·0	50·0	71·88	28·12	172	65·39
12	23,000	6 0	30·0	14·0	50·0	70·37	29·63	172	77·18
13	24,000	3·0	27·0	16 0	54 0	70·96	29·04	172	68·61
14	24,500	4·0	26·0	13·0	57·0	68·18	31·82	172	69·68
15	25,000	3·0	26·0	14 0	57·0	66·67	33·33	172	75·04
16	26,000	5 0	28·0	15·0	52·0	70·00	30 00	172	66·60
17	27,000	8 0	33·0	14·0	45·0	68·00	32·00	172	64·85

* Highest efficiency, grinding, and horsepower computed.

These tests were made to determine the most economical conditions under which the tube mills at our plant could be operated, taking into consideration the desired fine grinding as the end to be obtained, with the least consumption of power and the least wear on pebbles and linings, per ton of ore ground. These three points, namely, the fine grinding, the power consumption, and the wear on pebbles and linings, all seem to depend on the following four conditions of operation: (1) The pebble load; (2) the tonnage; (3) the thickness of pulp; (4) the speed of rotation. A change in any of the above will produce a change in two and usually all three of the points given, but the most noticeable effect will be seen in the power consumption. Of these four conditions we have control of the first three, but the last has usually to remain constant on account of the difficulty and expense of changing the speed of the drive, so its bearing on the question can be dispensed with.

While the following series of results were obtained from the operations of the mills at the Colorado plant, and the conditions prevailing here are radically different from conditions at most other plants, the general conclusions to be drawn from a comparison of the different tables given may be of interest to other operators. This will be especially true of the data given by Fig. I., showing the variations in horsepower due to different pebble loads. For although this power curve would vary with different ore feeds and different amounts of solution and pulp, the variation would be uniform and would in no way detract from its application. The other tables given would, of necessity, vary with the different practices, but a comparison of each one in relation to the others will give in a general way an idea of what might be accomplished.

TABLE II.—Showing per cent. ore and solution, fineness of grinding, and horsepower, with varying solution feed; pebble volume and ore feed remaining fixed.

Pebbles lb.	Ore Feed Inches.	Tons Ore per 24 Hours.	Tons Solution per 24 hours	On 60.	On 100.	On 150.	Thro'gh 150.	Ore percent	Solution percent	Indicated Horse-power.
20,000	3	172	68	12·5	36·0	10	41·5	71·43	28·57	45·0
20,000	3	172	75	13·0	34·0	12	41·0	69·56	30 44	48·9
20,000	3	172	90	8·0	30·0	13	49·0	65·67	34·33	55·8
20,000	3	172	92	8 0	32·0	14	46·0	65·20	34·80	57·4
20,000	3	172	98	9·0	33 0	12	46·0	63·78	36 22	58·0
20,000	3	172	111	8·0	33 0	13	46·0	60·70	39·30	56·9
20,000	3	172	113	7·0	31·0	12	50·0	60·44	39·56	55·0
20,000	3	172	136	8·0	34·0	12	46·0	55·71	44·29	55·8
20,000	3	172	196	7·0	32·0	14	47·0	47·10	52·90	59·0
20,000	3	172	207	5·5	30·5	13	51·0	45·40	54·60	62·3
20,000	3	172	268	8·0	32·0	12	48·0	38·90	61·10	62·3

The mills used in this test were of the trunnion type, 5 ft. in diameter and 23 ft. long, made by the Stearns Roger Manufacturing Co., of Denver. The trunnions were 13½ in. in diameter by 20 in. long, the boxes being water-jacketed. The mills have a cast-iron main gear, 8 ft. in diameter with 90 teeth, and are driven by a steel pinion having 17 teeth attached to the pinion shaft. The mills are arranged in pairs, each pair being driven by a single pinion shaft with the pinions at each end engaging the main gears of of the mills. The pinion shaft has a drive pulley 11 ft. in diameter with a 26 in. face, which is belted direct to a 150 h.p. Westinghouse induction motor. There is a friction clutch on each end of the pinion shaft between the drive pulley and the pinion, which clutch allows of either mill being thrown in or out of action at will. This large drive pulley also has a balance-wheel effect and is useful in aiding the motor to pick up the load when a mill is first thrown in. The mills were lined with 4 in. silex blocks, thus making the inside dimensions 4 ft. 4 in. × 22 ft. 4 in. The total volume was 350 cub. ft. and each held a maximum pebble load of 33,600 lb.

During the tests one mill was charged with a fixed weight of pebbles without any solution or ore feed, to provide a constant auxiliary load, so as to avoid the necessity of making any correction in the power readings due to a change in the efficiency of the motor at the different loads. During the tests the motor never had a load less than 60% and never more than 96% of its full load rating, so that the efficiency of the motor was practically constant throughout.

The first readings of the electrical instruments attached for the purpose of determining the power consumption were taken after the mill had been running 40 minutes, as it was found that this length of run was necessary to allow the ore, pebbles, and solution to become thoroughly mixed so as to give a constant reading. The second reading was taken 20 minutes after the first reading to provide a check on the first. In this way errors in metre readings were avoided. Also after the first series of tests was made and the irregularities of the power curve at the end of the tests were noted, the pebble charge was reduced in the mill from 27,000 lb. down to 18,000 lb. and the tests duplicated from this point up to 27,000 lb., so that two power readings were had on all tests, and from 18,000 lb. to 27,000 lb. two separate and distinct series of tests were made. The duplicate tests checked each other within one-third of 1%.

Each charge of pebbles was weighed before being put in the mill and the ore feed and solution was determined at frequent intervals to insure that they remained constant. The pulp flowing from the discharge end of the mill was sampled during the last 20 minutes of the test and from these samples the percentages of ore and solution, and the screen tests were made.

Three different series of tests were made. The first consisted of 17 tests in which the ore and solution feed remained constant while the pebble load was increased from zero to 27,000 lb.

The second consisted of 11 tests in which the pebble load and ore feed remained fixed while the solution feed was varied.

The third consisted of 8 tests in which the pebble load remained fixed and the ore and solution feed were made variable.

The results of these three series of tests are shown in Tables I., II., and III., respectively. In these tables a slight variation will be noted between the amount of ore and solution given as being fed to the mills and the amount as actually found in the pulp discharged from the mill. This is due to a varying amount of moisture in the ore fed to the mills, and to a difference in the flow of solution due to varying heads in the solution tanks constituting the source of supply. The ore in all tests consisted of tailings from the chlorination barrels, which had been previously crushed to pass a 12-mesh, No. 19 wire screen. In the accompanying tables, tons of sand and solution are approximated, from weight per 10 seconds, taken at end of each test, and slight variations are obvious, due to sand containing varying percentages of moisture, and solution flowing under varying heads.

TABLE III. Showing per cent. ore and solution, fineness of grinding, and horsepower, with variable ore and solution feeds, pebble volume remaining fixed.

Pebbles lb.	Feed Inches.	Tons Ore per 24 Hours.	On 60.	On 100.	On 150.	Thro'gh 150.	Ore percent	Solution percent	Indicated Horsepower.
20,000	3	172	7·0	32	13	48·0	64·71	35·29	56·40
20,000	3	172	13·0	35	11	41·0	66·67	33·33	54·28
20,000	3½	190	12·5	36	10	41·5	71·05	28·95	51·60
20,000	3½	190	14·0	34	12	40·0	67·86	32·14	54·80
20,000	4	216	16·0	34	14	36·0	68·18	31·82	53·20
20,000	4	216	14·0	36	16	34·0	69·70	30·30	49·40
20,000	4½	231	26·0	38	11	30·0	66·67	33·33	47·50
20,000	4½	231	30·0	30	10	30·0	72·22	27·78	43·50

In the test series shown by this table, a uniform feed of 172 tons of ore to 69 tons of solution per 24 hours was maintained.

From the results given in Table I. two curves were plotted, Fig I. showing the relations between the power consumed and the amount of pebble load, and Fig II. showing the relation between percentage of ore ground to 100 mesh and the amount of pebble load.

In Fig. I. the abscissae indicate pebble loads, and the ordinates the corresponding horsepower required for fine grinding. At the right of the figure is shown the depths to which the successive loads of pebbles fill the mill. The first test was taken when the mill contained no pebbles but was filled with ore and solution. The mill in this state took 10·2 h.p. The first charge of pebbles was 3,000 lb. and the horsepower required was 18·8. The next addition of 3,000 lb., making a total of 6,000 lb. of pebbles, required an increase of only 1·57 h p. and the addition of 3,000 lb. more required an increase of only 2·13 h.p. From this point the horsepower increased rapidly and uniformly with each addition of pebbles, until the first maximum of 77·18 h.p. was reached at a load of 23,000 lb. of pebbles. From this point on the horsepower decreased to 68·61 h.p. upon the addition of 1,000 lb. more of pebbles, but again increased to

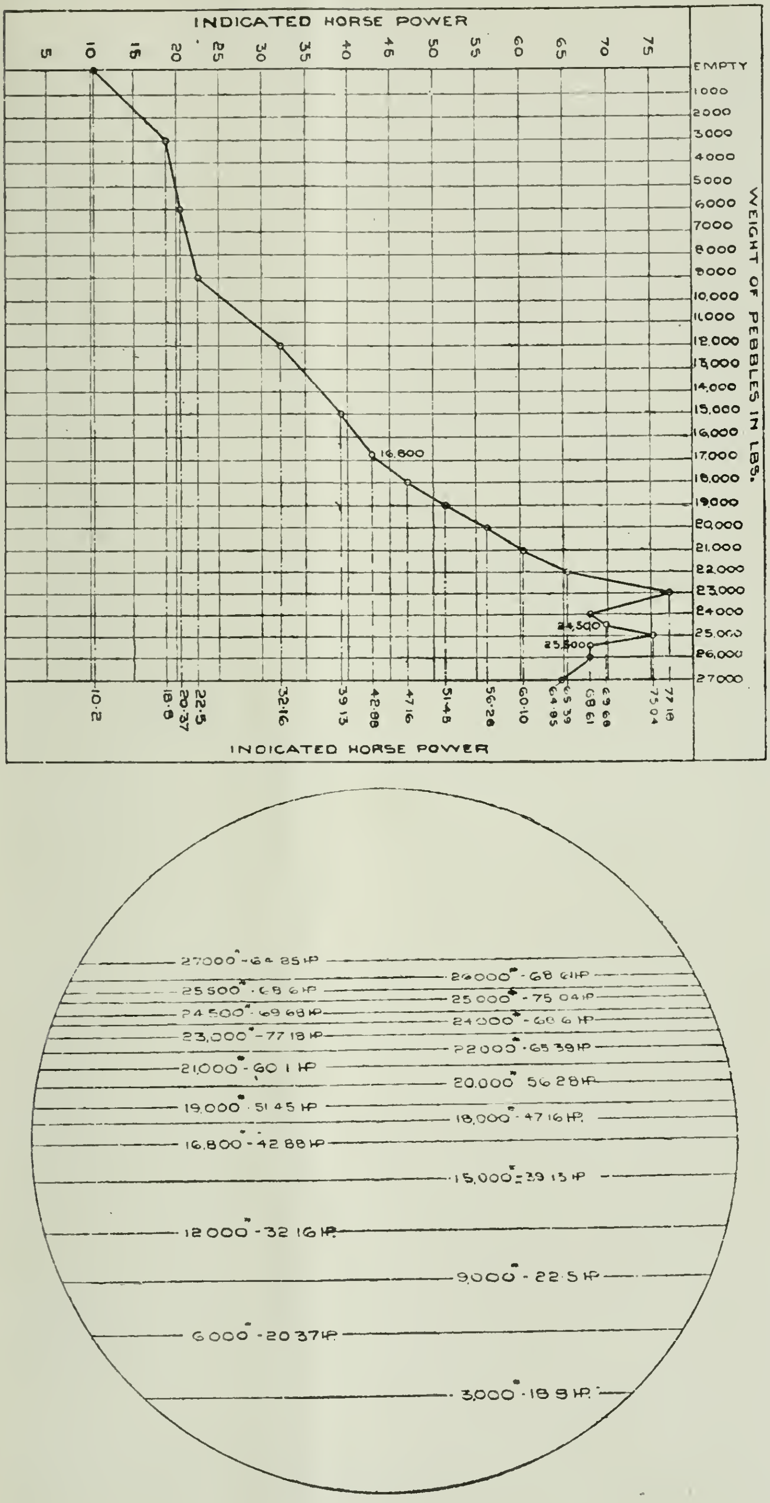


FIG. I.

69·68 h.p. upon the addition of 500 lb. additional pebbles. An addition of 500 lb. pebbles more increased the horsepower to the second maximum of 75·04 h.p. and from this point the successive additions of more pebbles resulted in a steady decrease of power required to the point shown, by the last reading of 64·85 h.p., when the mill contained 27,000 lb. of pebbles.

From the above it will be noted that when the mill contained 23,000 lb. of pebbles, or approximately 68·5% of its entire volume, the maximum horsepower was required; *i.e.*, 77·18 h.p. When the mill contained 24,000 lb. of pebbles, or approximately 71·4% of its entire volume, it required 68·61 h.p., which approximates a 11% decrease in horsepower by a 3% increase in volume of pebbles. When the mill contained 25,000 lb. of pebbles, or 74·4% of its volume it required 75·04 h.p., this being a 9·3% increase in power consumption from an increase of 4% in pebble volume. Another fact worthy of note is that the same horsepower consumption was required when the mill contained 71·4% and 77·3% of its entire pebble volume.

In Fig. II. the abscissae again represent the pebble loads but the ordinates represent the percentage of the ore ground to pass 100 mesh. Starting with 3,000 lb. of pebbles the mechanical or screen analysis of ground pulp showed 30% to pass 100 mesh. The same amount was found after the load was increased to 6,000 lb. At this point a uniform and steady increase in fine grinding was observed until 16,800 lb. of pebbles had been added to the mill. Here a sudden increase in the fine grinding was caused by the addition of 1,200 lb. more of pebbles which brought the amount of fines up to 67·5%, an increase of 21·5%. The addition of 1,000 lb. additional pebbles, making the total pebble load 19,000 lb., resulted in the small increase of only 0·5% in the fines. In the next test the adding of 1,000 lb. of pebbles resulted in the percentage of fines falling to 59% a sudden decrease of 9%. This decrease is undoubtedly due in part to the fact that at this point the percentage of ore to solution in the mill

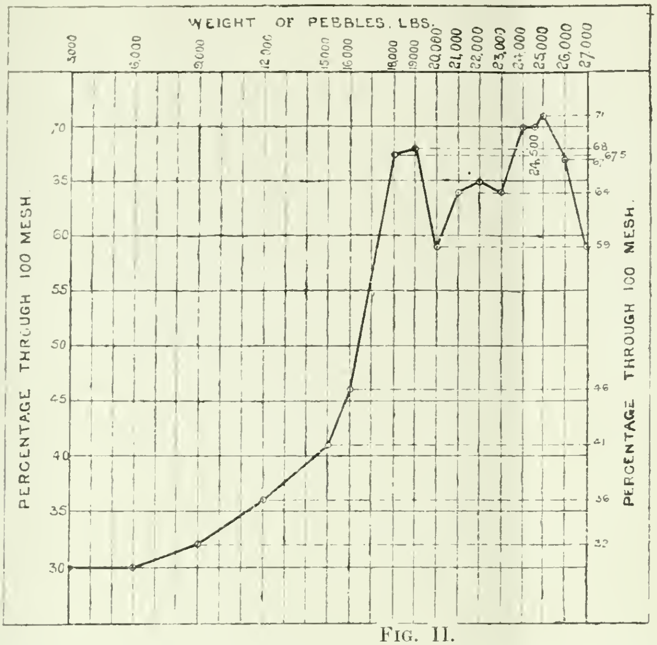


FIG. II.

% of sand in the solution during tests	
% through 100 mesh	% sand to solution
30	63
30	70
32	74
36	60
41	65
46	66
67.5	66
68	66
59	71
64	71
65	71
64	70
70	71
70	68
71	66
67	70

was inadvertently allowed to increase, as best seen in the table at the right of the figure. This was not noticed at the time and it was not possible afterwards to check this point. The table at the right of Fig. II shows the percentage of sand to solution at the time the different samples were taken. As the thickness of the pulp has more or less influence on the grinding, as is shown in Fig. IV., this table should be taken into consideration in explaining the irregularity of the curve from this point on. But in the opinion of the writer this variation in thickness of pulp does not account for all the irregularity in this curve. There is undoubtedly at this point, when the amount of pebble load varies from 18,000 to 25,000 lb , some condition prevailing in the tube that changes the motion of the pebbles and causes the marked variation in both the power consumption and the grinding effect. From a 20,000 lb. load the curve shows an increase in grinding with each addition of pebbles until 25,000 lb. have been added. From here on the grinding effect decreases regularly. It is interesting to note that when the mill contains 25,000 lb. of pebbles the maximum grinding effect and the second maximum in the power consumption are observed.

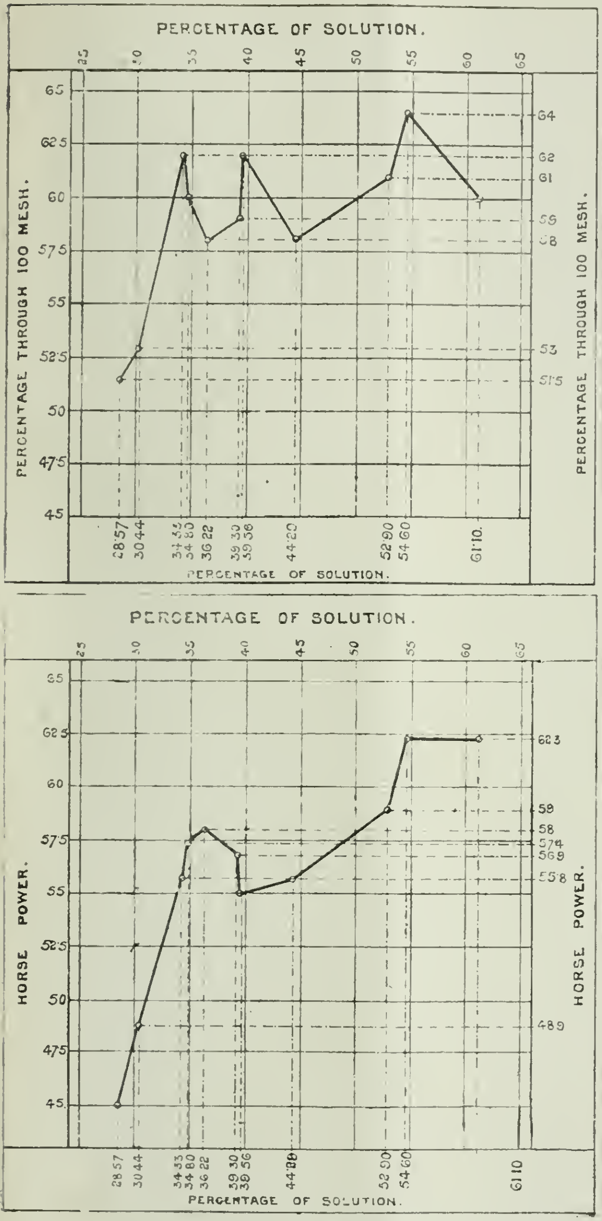
A comparison of Fig. I. and Fig. II. shows that the most efficient grinding, *i.e.*, the greatest amount of ore ground to pass 100 mesh for the lowest horsepower consumption, is shown with 19,000 lb. of pebbles. With 20,000 lb. of pebbles, 9% less sand screens 100 mesh. With 21,000 lb. pebbles a better grinding is shown but with a 7% increase in horsepower. With 23,000 lb. of pebbles a slight decrease in grinding is noted although at this point the highest horsepower consumption is registered. With 25,000 lb. of pebbles the finest grinding is obtained, but with a 45% increase in horsepower to get a 4·4% increase in grinding over that obtained at 19,000 lb. It is interesting to note that with 25,000 lb. of pebbles or 74·4% of the mill volume. the limit for filling the mill is reached, as far as grinding effect is concerned.

In this table the pebble load was constant at 20,000 lb., and the ore feed at 172 tons per 24 hours. From this table the curves in Figs. III. and IV. were plotted.

Figs. III. and IV. should be considered in their relations to each other. In both curves the percentages of solution to ore in the pulp are taken as the abscissae, but in Fig. III. the horsepower is taken as the ordinate, while in Fig. IV. the percentage of ore passing 100 mesh is taken as the ordinate. As shown in Fig. III. the first test was made when the percentage of solution to ore was 28·57%. A gradual increase in power was required as the percentage of solution was increased, until 36·22% was reached. From here the curve is more nearly straight, decreasing slightly as the pulp is made thinner and again increasing slightly above the first maximum when 54·60% of solution was present. This would tend to indicate that after 36% of solution was present the charge in the mill was thin enough to allow the pebbles to roll freely and subsequent additions of solution would not affect the power consumption to any great extent.

In Fig. IV. the irregularity of the curve is more marked but the difference between the several tests is not as great as it would seem at first glance, as after the first two tests the greatest difference made in the percentage of fines is only 6% from the maximum obtained with 54·6% solution. Comparing Figs. III. and IV., it would seem that after the percentage of solution in the pulp has reached about 35%, both the grinding and the power consumption stays reasonably uniform. Up to 35% of solution it would seem that the pulp charge in the mill is so thick that the free rolling of the pebbles is prevented, to the detriment of the grinding, although this condition favours the power consumption. If the curve in Fig. III. could be shown with a larger dilution of the pulp it would undoubtedly tend downward again after a certain dilution was reached. This conclusion is drawn from the fact that after the tests recorded above were taken, the ore feed was shut off and the solution was allowed to run into the mill for several hours to wash out the ore before taking out the excess of pebbles. When the mill was filled with only pebbles and solution a power reading was taken which showed that in this condition the mill took 11 h.p. less than the maximum when the ore feed was on.

In Table III. are recorded the results of a series tests taken to determine the horsepower required with different tonnages fed to the mill. Two tests at each feed were made, the solution being varied in each case. The mechanical analysis of this pulp from



Figs. III. and IV.

each test is also given so that the grinding effect can be noted.

It is interesting to see that at each separate feed the power was lowest at the least percentage of solution in the pulp. Also that the power shows a decrease from 56·4 h.p. with a feed of 172 tons of ore per 24 hours to 43·5 h.p. with a feed of 231 tons per 24 hours. But this decrease in horsepower due to a greater tonnage is more than counterbalanced by the decrease in the percentage of ore passing 100 mesh from 61% to 40%. In the thicker pulps, which show a less horsepower consumption, we also get a decrease in the percentage of fine ore. But an analysis of the results in reference to the amount of grinding work done by the mills, *i.e.*, the actual number of tons of ore ground to the different meshes, shows that at 100 mesh the number of tons of ore ground per horsepower is nearly constant at all feeds, while tons of ore per horsepower passing 60 mesh is larger with the heavier feeds. This last is probably due to the fact that at 60 mesh we get the benefit of the ore already ground to this mesh in the original ore and whose presence would be more pronounced with the larger amount of ore passing through the mill when the feed was heavier.

It is regretted that the wear on pebbles and linings under the conditions occurring during the various tests made could not be determined, but as this would necessitate a prolonged operation for each one it was impossible to do so. But experience has shown that the thicker the pulps can be carried the less the wear on pebbles and lining, and as from Figs. III. and IV. we note the effect of the condition of the pulp on the power and grinding some conclusions can be drawn as to the proper adjustments of solution and ore so as to give the best results.

In the above no attempt has been made to draw conclusions, except in a general way. Each test has been given in detail, so that other operators can arrive at such conclusion regarding the conditions of economic operations as may apply to their particular case."—H. W. Fox.—*Mines and Minerals*, June, 1908, p. 537. (A. McA. J.)

Protective Coatings for Cyanide Vats.—*Tar Varnish, for Coating Wood Vats.*—Heat 80 lb. of tar in a water bath at 160° F. for some time until all water evaporated, then add with constant stirring 80 lb. of Roman cement of hydraulic lime. The mixture should remain thin and fluid. Heat, and apply hot. No quicklime must be used. This varnish is acid-resisting and flexible.

Coating for Wood Vats.—Benzol, 3 parts; wax, 1 part; dammar or kauri gum, 2 parts. Paraffin may replace beeswax with advantage. Dissolve the dammar and wax in the benzol, and apply like a varnish.

To Close Leaks in Iron Tanks.—Asphaltum, turpentine. Make a saturated solution of the asphaltum in the turpentine. Apply hot to the dry iron. Give three or four coats. — W. H. Gaze. — *Practical Cyanide Operations*, p. 226. (W. A. C.)

Vanadium in Steel Castings.—"The addition of vanadium to ordinary steel for castings has much more than doubled, and in many cases more than trebled, the breaking strength of the locomotive side frames which are now being used by several American railway companies. According to the *American Machinist*, the Union Steel Castings Co., of Pittsburg, uses the open-hearth furnace for melting its steel, and the vanadium is added to the mixture just before tapping. The slag is raked from the top of the molten metal, the ferro-vanadium thrown in and the whole allowed to stand a few minutes, so that the vanadium will thoroughly mix with the metal. The furnace is then tapped and the moulds poured.

If the castings are properly annealed after being cast, the vanadium-steel castings are more easily machined. This in itself is quite an item, as one locomotive builder has had to pay nearly twice as much for machining side frames as it would cost to machine the ordinary carbon steel."—*Engineering Review*, June, 1908, p. 379. (A. R.)

MICROSCOPIC EXAMINATION OF ROASTED SULPHO-TELLURIDE ORES. — The author has determined that the best temperature for finishing the roasting of ore from the Wano mines, Jamestown, Colorado, is 1,000° F., and that they should be subjected to this heat for about 30 minutes. The microscopic examination of the gold panned from 10 lb. of the roasted ore showed that at 1,100° to 1,300° F., a relatively large amount of an incrustation of TeO_2 was observed on the gold where its reducing effect would be most obnoxious. At 1,000° F. this was very slightly the case. No figures are given for the differences shown in extraction but a very interesting account of the details of roasting effect is given.—W. H. DAVIS.—*Western Chemist and Metallurgist*, May, 1908, p. 137. (H. A. W.)

ORE CRUSHING. — The attainment of fullest economy in crushing will be found in a speed that is sufficiently high for tough ore, on amplitude that is adequate for elastic ore, and from a moderation of both speed and amplitude for brittle ore. Whenever the reduction in size is intended to be great, the fine must be protected from further crushing either by water-suspension, repeated sizing, or abundant opportunity for rearrangement of the compressed ore by the provision of interstitial space that accompanies uniform sizing. It depends upon producing the effect of crushing rather than of grinding, particularly upon coarse sizes, and of employing hard and non-elastic surfaces in contact with the ore. It demands solid foundations for machinery, care, cleanliness, and protection of bearings, tested lubricants, and the avoidance of all unnecessary irregularity of shape in high-speed running parts designed for the construction.—*New Zealand Mines Record*, April, 1908, p. 398. (A. L. E.)

THE LASZCZYNSKI COPPER PROCESS.—"This process has been described by K. I. Grivnaka in a Russian technical journal, and also by Prof. Piltschikoff in a paper read before the Faraday Society on April 28, 1908, and the following details are drawn from those two sources combined.

The Laszczynski process is based upon lixiviation of the roasted copper ore with a dilute solution of sulphuric acid, and electrolysis of the resultant solution, containing copper and iron sulphates and free acid, in special vats, using lead anodes and copper cathodes. The principle of the process is not new, having been tried experimentally both in Germany and America ; but Laszczynski has introduced some novel features, which are claimed to convert a process that has hitherto failed, into a practical and economical success. The ore is delivered from the mine in three grades—testing respectively 4·5%, 2·8%, and 1·5% of copper. This ore is carefully sorted to free it from gangue, etc., and is mixed and ground to pass through a one millimetre mesh sieve. The finely ground ore is then mixed with clay and formed into briquettes, which are roasted. This briquetting and roasting is one of the important features of the Laszczynski process, since by this treatment the iron is converted into an oxide that does not dissolve in the leaching liquor, and one of the difficulties met with in working other processes of this character is thereby avoided. The copper after the roasting operation is present in the roasted ore partly as sulphate and partly as oxide. The roasted ore is now leached with a 5% solution of sulphuric acid in lead-lined vats, worked upon the same principle as black-ash lixiviating vats, the saturated copper solution meeting the fresh ore, and the exhausted ore being leached with fresh acid solution. The liquor obtained by this leaching operation contains 5% of copper and 1% of free acid, and is filtered before passing into the electrolytic vats.

These are square wooden boxes lined with sheet lead and measuring 1 metre in each direction. Each vat when fully charged contains 9 anodes and 8 cathodes, the former of sheet lead, 3 mm. thick, and the latter of thin copper sheet. The anodes are encased in cotton cloth. The electrolyte is constantly agitated during the passage of the current, by aid of wooden frames placed between each anode and cathode, and operated by mechanical means. The current strength is 900 ampères per vat, and the e.m.f. required varies from 2 to 5 volts, while the yield is 25 kilos. per 24 hours. In the course of one month the cathode sheets attain a thickness of 20 to 30 mm. and increase in weight to 55 lb. The exhausted electrolyte contains 1 to 1½% of copper and 5% of sulphuric acid and is used again, with the addition of fresh acid, for leaching purposes. The presence of carbonate of lime in the ore leads to some loss of acid due to the formation of calcium sulphate, which is removed during filtration ; but this loss is partly compensated by the presence of sulphur in the raw ore and by the formation of copper sulphate during the roasting operation.

The copper deposited at the cathodes has the colour and appearance of fine electrolytic copper, and is slightly crystalline in structure.

The re-solution of copper at the cathodes which usually occurs when a mixed solution of iron and copper sulphate is electrolysed is avoided in this process by the use of the covered lead anodes. The use of linen covers for these, in some unexplained manner, prevents the formation of ferric sulphate by oxidation of the ferrous sulphate present in small amounts in the solution, and the full theoretical yield of 1·15 gm. of copper per ampère-hour is therefore obtained at the cathode. The cotton covering of the anodes is stated to last about four months without renewal.

The success of this extraction process at Boleslav has led to the erection of a larger works for operating it on the river Nid, not far from the original experimental installation of plant. The cost of a plant capable of producing 1,000 kilos. of copper per day is 27,000 roubles (equal to £4,300), and the process is stated to be equally efficient when worked on a small or large scale."—*Times Engineering Supplement*, May 10, 1908.—*Journal of the Society of Chemical Industry*, May 30, 1908, p. 510. (A. W.)

ACCUMULATION AND ABSORPTION OF GOLD AMALGAM ON COPPER PLATES.—"Accumulation means the gradual formation of a hard scale of amalgam on the surface of the plate, which cannot be removed by the usual rubber scrapers. 'Absorption' represents the amalgam that has actually been absorbed by, or has soaked into the plate, and which can only be recovered by melting down the plate. The writer details the accumulation occurring on three sets of 'apron' plates in Columbia during a period of six years. By comparing the weight of scale accumulated, with the total value of the ore that had passed over the plates, he calculates the average rate of accumulation to be 1·27 to 2·07 dwt. per ton of an ore averaging about 11 dwt. per ton." author entirely disagrees with the view that copper plates absorb a considerable amount of gold, sometimes estimated at 70 oz. per plate, even though they may have become saturated with mercury. The original plates from the above mill, which were $\frac{1}{16}$ in.

thick, were found to be completely saturated with mercury, but when broken up, only yielded 4·243 oz. of fine gold. Other details are given showing that only 8·96 oz., 8 oz., and 9 oz. had been found in plates that had been in use for 4, 10 and 2½ years respectively. A paper on the same subject by W. F. A. Thomae is in agreement with the above."—E. HALSE, *Transactions Institute of Mining and Metallurgy*, May, 1908.—*Journal of the Society of Chemical Industry*, June 15, 1908, p. 571. (A. W.)

IRON ORE FROM NATAL.—A sample from a surface outcrop at Sweetwaters, near the railway, about 1 lb. in weight and consisting of fragments of brownish limonitic ore without visible quartz or other impurity gave on analysis:—

Fe_2O_3	71·31%	equivalent to	49·91% Fe
FeO	—		
MnO	2·36	,,	1·82% Mn
SiO_2	6 08		
P_2O_5	0·7	,,	0·3% P
S	—		

—*Bulletin Imperial Institute*, vol. vi., No. 1, p. 27. (J. A. W.)

MINING.

RE-CAPPING WINDING ROPES.—"In re-capping a rope, which should be done three or four times a year, says George H. Winstanley, lecturer in mining at Manchester University, a suitable length of rope should be cut off with the old capping, a length not less than one-half of the circumference of the headgear pulleys.

There are certain points in the cycle of operations in winding which result in greater wear in the corresponding places on the rope than in other parts. For example, that portion of the rope which is in contact with the headgear pulley when the cage is at the top and the bottom of the shaft respectively. The starting of the wind, and the manipulation of the engine to facilitate the loading and unloading of the various decks, induces greater frictional wear at these points.

Another point where wear takes place, especially in quick winding, corresponds with the point where the speed of the engine is checked, the point where the brake is applied to the drum, and the rope in turn acts as a brake to check the revolution of the headgear pulley.

For these and other reasons it will be seen that only by cutting off a proper length of rope at each re-capping can the excessive wear at these points be prevented from becoming too local. The piece cut off should be carefully examined and tested."—*The Mining World*, April 11, 1908, p. 596. (W. A. C.)

MINE TIMBER PRESERVATION.—"The results of experiments in preservation of timber have been published by Mr. E. Henry. They were made in the iron mines, near Nancy, where the gallery props are covered with various kinds of fungi. The wood experimented with was laid on the gallery floor and on the mouldy props. The increase in weight of five kinds of wood—viz., oak, fir, beech, poplar, Aleppo pine, injected with carbolineum, was about the same after three years in the dry galleries or outside in the rain. This demonstrates that carbolineum is not easily dissolved. The wood injected during 24 hours with carbolineum or tar, or microsol, was intact at the end of three years. Tar, however, is the least reliable of these preservatives, for when the wood is broken anywhere on the surface it has no protection against the agents of destruction. Hydrofluoric acid and antinnonine are useless for wood exposed to the inclemency of the weather, but they preserve wood under cover. Hydrofluoric acid prevented the alburnum of oak planks from decomposing. Lysol is as objectionable inside as out. Further details are given in the *Revue Scientifique* of April 4.—*Revue Scientifique.—London Mining Journal*, April 25, 1908, p. 517. (A. R.)

MINE TIMBER PRESERVATION.—"According to the director of the Stelns mines (Mulheim) all the processes for preserving wood which involve dessication by heat are not to be recommended as they make the wood brittle. Impregnation with a saline solution, on the contrary, gives good results. The wood thus treated is very durable (some specimens are ten years old). Moreover, there is no danger of fire, and the air in the mine is cooled instead of being vitiated. The wood, of course, must be quite dry. Wood thus treated, after being five years in the mines, was in a perfect state of preservation."—*Echo des Mines*, June 15.—*London Mining Journal*, June 20, 1908, p. 754. (A. R.)

COAL DUST EXPLOSIONS.—"Everything relating to dust in mines, cause of the recent catastrophies of Courrières and Monongah, is of great interest to engineers. For this reason the study made by Mr. Dautriche, engineer of the Government Explosives Works, the results of which were recently presented to the French Academy of Sciences, is of much interest. Mr. Dautriche's experiments demonstrate that to prevent combustion of the products from detonation of explosives with partial combustion, it suffices to add small quantities of potassium or sodium salts. Employment of these explosives in fiery mines should not then be rejected *a priori*. On the other hand, the other substances added, and especially the alkalino-terreous salts, have only a limited action on combustion of the products. According to the heat disengaged they seem only to oppose combustion of the carbon monoxide. Addition of alkaline salts must also be made to avoid ignition of the combustible substances which pre-exist in vicinity of the cartridges (firedamp, coal dust). The safety of French explosives with base of nitroglycerine, nitronaphthaline, and ammonium nitrate would be much increased by addition of potassium or sodium nitrate. Replacement of a small quantity of ammonium nitrate by an equivalent amount of nitre has scarcely any effect upon the temperature of detonation. Thus from the safety point of view nitre has produced the same effect as a decrease of 300° C. in temperatures. Briefly, says the author, a cloud formed by dust from any potassium or sodium salt impedes combustion of gas and combustible dust."—*Echo des Mines*, March 26.—*London Mining Journal*, April 18, 1908, p. 477. (A. R.)

ACCIDENTS IN WINDING.—"The breaking of a colliery winding rope, whenever it does occur, is more likely to be the result than the cause of an accident.

The adoption of the so-called safety cage is not the remedy. Safety cages do not prevent serious strains upon the rope, neither do they insure against neglect. In good British colliery practice it has become almost universal to allow a factor of safety of ten for winding ropes. The recent Transvaal commission considered this question, and appears to have come to the conclusion that a factor of safety of six was a sufficiently liberal allowance. The Transvaal commission may be right in its conclusion, at least so far as conditions in South Africa are concerned, but the writer would be sorry to think that a similar conclusion would be arrived at in this country, or that the more liberal figure of ten were to be reduced.

In speaking of the factor of safety of ten, it should be clearly understood that the intention is to provide at the outset a rope which shall have a strength of not less than ten times the gross load to be raised, this gross load being made up of the weight of the cage, the tubs with their full weight of coal, and the weight of the rope itself. The steel wire used in the manufacture of the best qualities of winding ropes has a breaking strength of from 100 to 120 tons per square inch of sectional area. The writer offers for what it is worth a simple rule by which the weight in pounds per yard, and the size of a winding rope suited for a given load and depth of shaft, can be calculated with fairly accurate results. It must, however, be understood to refer only to round wire ropes of the usual construction, made of high-grade wire having a tensile strength of about 120 tons per square inch. It is based upon the fact that in round numbers the weight of 20,000 yards of such a rope, of any size, is practically equal to its breaking strength, and that the weight of 2,000 yards therefore represents its safe working load. The rule is as follows:—

Take the maximum suspended load in pounds (that is, the total weight of the loaded cage) and divide by the difference between 2,000 and the depth of the shaft in yards. Thus, take a load of 8 tons, and a shaft 600 yards deep:

$$\frac{8 \times 2.240}{2,000 - 600} = \frac{17,920}{1,400} = \text{say, } 12\tfrac{3}{4} \text{ lb. per yard.}$$

The nearest rope to this, in the list of a well-known manufacturer, under the head of 'improved plough steel' is one weighing 25 lb. per fathom, of which the breaking strength is stated to be 119 tons; the working load will therefore be 11·9 tons. Six hundred yards, at 25 lb. per fathom, amounts to 3·35 tons, giving a gross load of 11·35 tons, showing that the rule in this case gives a correct result. The square root of the weight in pounds per fathom is equal to the circumference of the rope in inches. When a new rope is purchased, the maker should be required to furnish a certificate of test, both of the wire used and a sample length of the finished rope. The tests should be of as comprehensive a character as possible, embracing tensile, torsional, bending, and fatigue tests; and all the details should be fully set forth on the certificate.

The demand for so-called cheap ropes—that is to say, ropes which can be bought at a low figure per ton, but which in practice prove really to be costly on account of their comparatively short life—has led to the manufacture of ropes that can be purchased at a price which, if the purchaser would only reflect for a moment, is very little higher than the cost of best quality steel bar, and therefore leaves nothing for the various processes intervening between the raw material and the finished rope.

A rope may be manufactured from wire which has cost much less to manufacture than another wire, and yet the two samples of wire will satisfy the same tests as to tensile strength, torsion and bending, but not fatigue. Vaughan & Epton have invented an appliance—illustrated in the report of the Transvaal commission—which enables the fatigue test to be applied. Briefly, the object of this machine is to enable a wire to be put under load, say, one-fourth of its tensile strength, and worked continuously around pulleys, bending in opposite directions, until fracture takes place. The pulleys, it must be remarked, are not small relatively to the diameter of the wire, but of ample diameter, so that the cause of final fracture is actually fatigue due to working stresses, and quite distinct from the ordinary bending test."—Geo. H. Winstanley.—*Mining World*, May 2, 1908, p. 720. (H. A. W.)

Determining Depth of Water in Boreholes.—"A new process of determining the depth of water in a bored well, while pumping is going on, is described by the author. A known length of small pipe (¼ in. diameter) is lowered into the water. This is connected to a pressure gauge, graduated to pounds. Then air is blown into the pipe and, when the needle on the gauge becomes quiet, the pressure is read. This indicates the pressure necessary to sustain the column of water above the end of the submerged pipe; therefore this depth of submergence is obtained by reducing the pressure, indicated by the guage, to feet of head. The height of water surface in the well is then easily determined."—C. B. Burdick, *Journal of the Western Society of Engineers*, Dec., 1907.—*Engineering and Mining Journal*, March 21, 1908, p. 607. (G. H. S.)

Deep Mining.—"The Morro Velho mine is the deepest gold mine in the world. It has been worked by the St. John del Rey Mining Company, Ltd., since 1830. In 1868 the timbering of the old mine was burnt out, and in 1886 the mine caved in. Under the present superintendent, Mr. Chalmers, the mine was re-opened in 1889, and, in spite of great difficulties and expenses, work has proceeded until the lode has been intersected at a depth of 4,264 ft. Among many interesting features are the pneumatic hammer drills used in the place of boring, and the Chalmers-Wilder oxygen process, which extracts 70 to 80% of the bullion from the ore in four hours."—*London Mining Journal*, May 2, 1908, p. 532. (A. R.)

Dust Collector for Rock Drills.—"At a recent session of the Société de l'Industrié Minérale, M. Renmaux gave a number of suggestions as to the use of rock drills, and, among other novelties, described an easily made apparatus for catching the dust as it comes from the drill hole, thus avoiding the use of a hollow drill with spray of water. He objects to the latter method of preventing contamination of the air because the mud and the dampness are almost as disagreeable as the dust would be.

His apparatus consists of a conical wire frame, which can be made of 3 mm. steel wire, by coiling it on a wooden mandrel; the resulting coil is covered with flexible leather. The small end of the cone is fastened to the forward part of the drill cylinder. The large end has a heavy leather washer, and a tight contact between the washer and the rock wall is made with clay. The air escaping around the drill piston is sufficient to carry the dust off through the discharge at the back end of the cone, leading into a pail of water.

The dust collector has a length of 75 cm. and a diameter of 5·5 cm. at the small end and 10 cm. at the large end. Only this one size is required, for a bit will usually get dull, and will be replaced by a longer bit after every advance of 50 cm., which is within the range of elasticity of the dust collect ”—*Engineering and Mining Journal*, May 9, 1 p. 957. (G. H. S.)

Phosphate Deposits of Algeria and Tunis.—The deposits of North Africa are generally found at the junction between the chalk and tertiary systems, but those in the chalk are nowhere worth working. The principal mines in Algeria are (1) at Oran, where the phosphate rock occurs in pockets covered with an outer crust of breccia containing bones of recent mammals. Best deposits contain 60—82% $Ca_3(PO_4)_2$; (2) At Dra-El-Abiod, containing 20—45% $Ca_3(PO_4)_2$, and 12,000,000—14,000,000 tons are estimated in sight; (3) at Dye, where the phosphate bed is 3 metres thick, and contains 55—69% $Ca_3(PO_4)_2$; (4) at Knif, where 6,000,000 tons, averaging 56—65% are available.

In Tunis three districts are worked:—(1) At Beja four beds occur, 2—3 ft. thickness of poor quality; (2) at Meltani the deposits contain 5,000,000 tons, averaging 59·5—61·5% $Ca_3(PO_4)_2$; (3) at Redeyeff 12,000,000 tons, averaging 64·5% $Ca_3(PO_4)_2$.

In 1906 total exports of rock phosphate from Algeria and Tunis amounted to 1,049,865 metric tons (2,204 lb.) of which Tunis was responsible for 747,303 tons.—*Bulletin Imperial Institute*, vol. vi., No. 1, p. 87. (J. A. W.)

MISCELLANEOUS.

Spontaneous Combustion of Coal.—"Experiments made by Erdmann and Stolzenberg with ozone-bearing air prove that brown coal in the presence of water can easily be brought to spontaneous combustion. Marcasite has no influence. Spontaneous combustion is probably caused by the evaporation of the water at the surface of the brown coal forming ozone, which is at once absorbed by the humic acid. This causes an increase in temperature, increased evaporation, and increased formation of ozone. In this way is explained the circumstance that on sunny days after wet weather spontaneous combustion occurs. Moreover, sea-air is rich in ozone, and, consequently, ships' cargoes are liable to spontaneous combustion."—*London Mining Journal*, May 23, 1908, p. 624. (A. R.)

Alkali Destroys Cement.—"That ordinary white alkali of Montana attacks and destroys cement concrete and probably some kinds of sandstone, is one of the definite and highly important conclusions at which the department of engineering and chemistry of the state experiment station have arrived thus far in the investigations which they are carrying on in regard to alkali. The importance of this discovery is recognised when one considers the important part which cement plays in the construction of foundations, drains, sewers, irrigation work and other construction, and the fact that nearly all the soil of Montana contains more or less alkali.

This means that it is unsafe to use cement in any foundation or other structure coming in contact with the soil, in the alkali regions of the state, unless some method is devised to protect it from the action of the chemical. To devise some protection against this action of alkali on cement is one of the main objects of the further investigation of this subject which the experiment station expects to carry on.

The attention of the station was first called to this matter about a year ago by a letter from City Engineer C. W. Swearingen of Great Falls, to Prof. E. T. Tannatt, head of the engineering department of the station, asking him to investigate the cause of the disintegration of certain concrete sewers in Great Falls. Last fall, Prof. Tannatt and Edmund Burke, experiment station chemist, made a thorough investigation of the sewers referred to by Mr. Swearingen, and also of some concrete and sandstone foundations in the same city. These sewers, which had only been in a few years, were found to be in many places nearly destroyed by the action of the alkali in the soil surrounding them, and the sandstone foundation of the court house at Great Falls was found to be affected both on the outside and on the inside, though not seriously damaged. A similar action has been noticed on cement concrete in other parts of the state.

The experiment station is publishing a bulletin on the subject which describes the work done up to this time in this important investigation. In this bulletin it is recommended that neither cement concrete nor sandstone be used where it will come in contact with moist alkali soil, and that where concrete or sandstone has already been placed in alkali soil, the work be thoroughly cleaned and protected from the action of the alkali, for a time at least, by a coat of asphalte."—*Pacific Builder and Engineer.*—*Mining Science*, May 14, 1908, p. 464. (A. R.)

Sunday Work.—The question of Sunday work on the dredges has been discussed by the Southland Presbytery, which issued a circular to the various dredgeowners calling attention to the desecration of the Sabbath that was going on. A number of replies were received, stating that no work was done except under very exceptional circumstances, and satisfaction was expressed by the Presbytery that on several of the dredges no work was done on Sundays. The Te Aroha Syndicate stated that at no time had dredging operations been carried on during the Sabbath day. All their managers were anxious to avoid Sunday work. The mines' department would grant a permit to work on Sundays only when there was danger to life or property, and these permits had to be applied for weekly; but they were very rarely asked for.—*New Zealand Mines Record*, April, 1908, p. 408. (A. L. E.)

Deterioration of Coal by Weathering.—"The weathering of coal, generally known as shrinkage, is a subject which gives rise to many discussions, but is not fully understood. Claims charging short weighing, and frequently suits, result when the purchaser of a car of coal finds that he does not receive as much coal as his invoice describes. For the benefit of dealers and coal consumers, the following may be of interest, especially since practical tests were made to verify the theories.

The causes for shrinkage of coal weights are as follows: (1) The evaporation of moisture or water in the coal. (2) The loss of volatile gases, which, combining with the oxygen in the atmosphere, cause a shrinkage in weight and loss of heat units.

By chemistry, high-grade bituminous coal is composed of 4% moisture, 35% volatile matter, 55% fixed carbon and 6% ash. This means that every ton of coal contains 80 lb. of water, 700 lb. gassy matter, 1,100 lb. carbon and 120 lb. earthy matter or ash.

When coal is freshly mined it usually absorbs more or less moisture from the air and water in the mine. Upon exposure to the atmosphere, and more especially during transportation, evaporation takes place, depending upon the climatic and weather conditions. This fact is generally known, but few coal consumers realise that in addition to the evaporation of the water the oxidation of the more volatile gases in the coal takes place and results in a much larger decrease in weight and heat values than is due to the evaporation of water.

Some interesting experiments have been made by the University of Illinois on this subject, showing a loss of 1,350 British thermal units per pound of combustible by storing coal for seven months with outdoor exposure, and the writer has ascertained that coal originally containing 2½% of moisture, upon exposure to the atmosphere for six months, lost 3% in weight, and yet the analysis indicated over 2% of moisture in the sample. This indicates clearly the shrinkage of the volatile gases. Had this coal been handled or transported the shrinkage would perhaps have been greater.

Coal stored under water for six months showed a gain of 3% owing to the absorption of moisture and prevention of slacking due to the water seal.

The chemical process is more apparent in the drier climates than it is in the lower sections where the ratio of humidity is greater. This condition may be compared with the evaporation or shrinkage of the higher volatile oils, such as gasoline and benzine, and the lesser evaporation of heavier oils, such as crude petroleum. The higher the ratio of volatile matter, therefore, the greater the shrinkage.

The shipper of coal can only base his selling price upon the weights ascertained at the mines. The farther the consumer is from the mines the greater will be the shrinkage, and consequently the retailing price must increase with the shrinkage. If the selling price at the mines were based upon delivery of a stated quantity it would of necessity be higher to the consumer near the mines and he would then lose the benefit of his proximity to the producing field.

The only place properly to account for shrinkage is at the point of consumption, and the retailing price should vary in each locality according to the distance from the mines and the general weather and climatic conditions.

This feature also applies to freight rates ; as coal is hauled greater distances the shrinkage increases. Freight rates are based on weights ascertained at the mines and contemplate a shrinkage in transit beyond the control of the carrier. Rates for longer distances are lower per ton-mile haul than those for shorter distances. Coal loaded in box cars shrinks less than that in open or stock cars where large surfaces are exposed to the drying action of the atmosphere and air currents." — Carl Scholtz. *The Mining World*, June 6, 1908, p. 914. (W. A. C.)

The M. and M. S. of A.—As so many of our members are connected with American technical societies it gives us particular pleasure in welcoming the accession to their ranks of this very promising new Mining and Metallurgical Society of America. We feel assured that not only the new but also the old world must derive considerable benefit from its institution, and wish them a continuous success in the achievement of their high aims. We append a leader on this subject from the *Mining World* of May 9, 1908.

" In drafting the constitution and bye-laws of the new Mining and Metallurgical Society of America, unusual care has been taken so that there may be no misunderstanding with regard to the objects of the association and the qualifications for membership.

According to the constitution adopted at a meeting held in New York on April 20 last, the association has for its objects the conservation of mineral resources, the advancement of mining and metallurgical industries, the better protection of mine investors and mine workers, the increase of scientific knowledge, and the encouragement of high professional ideals.

It is also specified that no member or officer shall receive salary, compensation, or emolument unless authorised by the bye-laws, or by concurring vote of two-thirds of the executive council. Members residing for a year or more beyond the limits of the United States. Canada and Mexico, shall not be entitled to vote nor hold office during the period of such residence.

Candidates for membership must have had eight years practical or professional experience, including not less than five years in positions of responsibility, in mining or allied lines of work. Graduates of approved engineering schools shall be credited with one-half the time prescribed for graduation.

The initiation fee shall be \$25 after the total membership reaches 250. The annual dues are \$10. The council may in its discretion permit any member not in arrears to become a life member on payment of an adequate sum for the purpose based on his expectation of life according to reliable tables of mortality.

The executive council, which shall manage the affairs of the association, is made up of the following men whose qualifications assure success to the new movement : Robert H. Richards, Boston ; Waldemar Lindgren, Washington, D.C. ; Joseph Hyde Pratt, Chapel Hill, N.C. ; T. A. Rickard and S. B. Christy, San Francisco ; W. R. Ingalls, H. S. Munroe, and J. R. Finlay, New York : F. Lynwood Garrison and W. A. Lathrop, Philadelphia ; L. S. Austin, Houghton, Mich. ; C. R. Claghorn, Tacoma, Wash. ; H. B. Patton, Golden, Colo. ; C. B. Dudley, Altoona, Pa., and George S. Rice, Chicago." — *Mining World*, May 9, 1908, p. 748. (H. A. W.)

The Carat.—"The International Committee of Weights and Measures has pronounced in favour of making the value of the carat uniform. The carat, the unit of quantity for precious stones, at present varies in different countries, and according to the decision of the committee it is fixed at 200 mgm. In conformity with this decision, the French Government is introducing a Bill whereby the designation of 'metric carat' in transactions relating to diamonds, pearls of fine quality and precious stones can be given to the double decigramme. The use of the word 'carat' to designate any other weight is prohibited." — *The Mining Journal*, June 13, 1908, p. 717. (H. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

Technical Methods of Ore Analysis. By Albert H. Low. Third edition. Partly re-written, 1908. 12s. 6d. (New York City, U.S.A. : John Wiley & Sons. London : Chapman & Hall, Ltd.)

" The third edition of this valuable work—the first issue appeared only three years ago—will be found to have maintained the high character which has resulted in the recognition of Mr. Low's book as a standard both in America and elsewhere.

Written mainly on 'Ore Analysis,' it covers a much wider field, and includes, in addition to most of the commercial metalliferous minerals and many of the more or less rare metals, much relating to fluxes, slags, furnace products, etc., and has useful sections dealing with fuels, boiler water, etc.

Although dealing chiefly with rapid methods the processes described have been carefully selected and are by no means rough. The necessities of the times have resulted in the evolution of improved processes or the introduction of new methods in which both speed and accuracy are combined, and the author has only included such methods as have been found sound and reliable.

Mr. Low has been fortunate in the co-operation of other analysts, who have given him the benefit of their own experience, and thus added enormously to the value of the work as a thoroughly up-to-date exposition of modern practice."—*London Mining Journal*, May 2, 1908, p. 537. (A. R.)

BURNING OF COAL WITHOUT SMOKE IN BOILER PLANTS. A Preliminary Report by D. T. RANDALL. (Washington, U.S.A.: Geological Survey.)

THE PURCHASE OF COAL UNDER GOVERNMENT AND COMMERCIAL SPECIFICATIONS ON THE BASIS OF ITS HEATING VALUE. With Analyses of Coal delivered under Government Contracts. By D. T. RANDALL. (Washington, U.S.A.: Geological Survey.)

"The author of these pamphlets, being the engineer in charge of the smoke abatement investigations, is an expert whose remarks command respectful attention. Mr. Randall states it has been found that generally, generating gas for use in a gas engine furnishes power with greater economy than is usual in steam plants. It is impossible in our space to quote, but we can promise that the author's remarks on city ordinances for smoke prevention, the principles of smokeless combustion, proper furnace equipment, and the difficulty in securing perfect combustion will afford most valuable and reliable hints to those interested in a subject of vital importance in English cities. Indirect methods of smoke abatements are discussed, and a useful list of books for reference is added.

The second work on our list—'Purchase of Coal under Government'—comments on the fact that until recently there has been little reliable information regarding the character of the coal supply of the United States. In 1904 a system was commenced of fuel testing and analysing, and the pamphlet before us summarises the information on the subject, as obtained by the Survey. Mr. Randall dwells on the advantages of definite specifications in the purchase of coal and the value of coal as a fuel. By direction of the United States Government the heating value in the statistics is expressed 'in British thermal units of coal containing approximately the same percentage of ash,' which 'is essentially a direct measure of value to the purchaser.' The methods of sampling and testing are set out accurately, and this is supplemented by a reference to useful works on these questions."—*London Mining Journal*, June 6, 1908, p. 698. (W. A. C.)

ANALYSIS OF ASHES AND ALLOYS. By L. PARRY, A.R.S.M. 5s. (London: *The Mining Journal*.)

"The first ten chapters deal exhaustively with the separation and determination of individual elements generally met with in the materials with which the book deals. It will be seen that Mr. Parry is, generally speaking, a convinced opponent of the 'dry assay,' excepting within certain narrow and well-defined limits, and certainly not for use in the buying and selling of metal contents of the various materials. His reasons for this attitude, and also for his deviation from many of the accepted methods of wet analysis seem to be perfectly sound; and it should be an easy matter for any chemist to satisfy himself that the conclusions arrived at are based on solid fact and not mere conjecture or opinion. Chapters XI. and XII., on these points, are especially valuable.

The second section of the book deals with the analysis of 'ashes,' etc., and we have here a very complete and useful summary of the various waste products and bye-products with which the metallurgical chemist has to deal, and many of which present great difficulty to the analyst.

In dealing with alloys, Mr. Parry points out the value of a systematic examination of the physical properties, and a preliminary qualitative chemical examination, as affording much useful information as to the character and scope of the succeeding quantitative examination, and the resulting saving of time and the possibility of falling into error.

Section III. of the work is devoted to miscellaneous analysis, and here, again, is much useful information, wonderfully compact and 'getatable.'

Finally, we have a very useful warning of 'errors of analysis' and a collection of miscellaneous data." *London Mining Journal*, July 11, 1908, p. 66. (W. A. C.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 275/08. Grant Bowen Holmes (1), Arthur Dunscombe Allen (2). Improvements in trolley heads for electric traction. 10.7.08.

(P.) 276/08. Arthur Walter Stuckey (1), Charles Hastings Snow (2). Device for recovering the metallic contents of such matter as alluvial deposit. 10.7.08.

(P.) 278/08. Alfred Charles Holtby. Improvements in means or devices for removing particles of iron or other magnetic bodies from crushed ore products. 15.7.08.

(P.) 279/08. Eustace Moriarty Weston. Improvements in rock drill steels. 15.7.08.

(P.) 280/08. Herman Lobel (1), Arthur Bath (2). Improvements in acetylene gas generator and lamps. 15.7.08.

(P.) 281/08. Allan Ewen Victor Fraser. Automatic petrolene gas generator. 17.7.08.

(P.) 282/08. John Hudson Hughes. Improvements in or relating to the stamps of stamp mills. 20.7.08.

(P.) 283/08. Thomas Hind Jackson. Improvements in apparatus for separating comminuted ores from liquids. 20.7.08.

(C.) 284/08. Wilhelm Alexander Bleeck. Improvements in primary batteries. 21.7.08.

(P.) 285/08. Karl Ruschenbaum. Improvements in rock drilling machines. 21.7.08.

(C.) 286/08. John Delleridge. Improved air compressor, usable also as a pump. 21.7.08.

(P.) 287/08. Otto Tennesen. Improved method and instrument for measuring and plotting stopes and such like. 21.7.08.

(P.) 289/08. William George Gibson. Improvements in securing gold and other precious metals. 23.7.08.

(P.) 290/08. Henry Samuel Potter. Improvements in means for withdrawing rock drills from bore holes. 22.7.08.

(P.) 291/08. S. J. P. Eloff (1). Theo. Senekal (2). Boring machine for deep boring. 23.7.08.

(C.) 292/08. William Thomas Smith. Improvements in wheels for motor cars and other vehicles. 24.7.08.

(P.) 293/08. John Penhale. Improved solvent for precious metals. 24.7.08.

(P.) 294/08. Alexander John Arbuckle (1), Alfred Osborne (2). Improvements in or relating to apparatus for separating comminuted ores from liquids. 27.7.08.

(P.) 295/08. Gustavus Adolphus Sheeley. Improvements in linings of tube mills and the like. 27.7.08.

(P.) 296/08. Gustavus Adolphus Sheeley. A new or improved cement or material for securing in position the segmental linings of tube and like mills. 27.7.08.

(P.) 297/08. Richard Carl Harris (1). John I'Ons (2). Improvements in the mortar boxes of stamp mills. 5.8.08.

(P.) 299/08. George Adie Scott. Improvements in pipes for conveying liquids. 29.7.08.

(P.) 300/08. James Inglis. Improvements in automatic tipping apparatus for mine trucks and the like. 30.7.08.

(P.) 301/08. Frederick Schiller. Improvements in acetylene gas generators or lamps. 30.7.08.

(C.) 302/08. Horace Sanderson (1). Rupert John Isaacson (2). Henry St. John Sanderson (3), Edwin Wardle (4), John Edwin Firth (5). Charles Earnest Charlesworth (6). Improvements in valve gears for steam and other fluid pressure engines. 31.7.08.

(C.) 303/08. Raymond Damseaux. Improvements in electric multiple intensity incandescent lamps and sockets for same. 31.7.08.

(C.) 304/08. Henry Byron Johnson. An improved jockey for endless wire haulage. 31.7.08.

(C.) 305/08. Richard Middleton Simpson. Improved apparatus for separating metals and the like from extraneous matter. 1.8.08.

(P.) 307/08. Alexander John Arbuckle (1), Alfred Osborne (2). Improvements in means for separating crushed ore products from liquids. 4.8.08.

(C.) 308/08. William Foden (1), The United Kingdom Self-Adjusting Anti-Friction Metallic Syndicate, Ltd. (2). Improvements in metallic packings for piston rods and the like. 5.8.08.

(P.) 310/08. Frederick Hudson Shephard Shepherd. An improved system or method of drilling or boring holes in mining and like operations and means therefor. 7.8.08.

(P.) 311/08. Walter Neal. Improvements in tube or ball mills. 7.8.08.

(C.) 312/08. Charles Christiansen. Improvements relating to pneumatic rock drills. 7.8.08.

(C.) 313/08. Harry Bentley. Improvements in trough conveyors, jigging screens or the like. 7.8.08.

(P.) 315/08. Gustavus Adolphus Sheeley. Improvements in liners for tube mills and the like. 10.8.08.

(C.) 316/08. Bertram Edward Dunbar Kilburn. Improvements in explosives. 11.8.08.

(P.) 317/08. Gustav Theodor Falke. A new or improved fluid for preventing or removing incrustation or scale in steam boilers or condensers and the like. 12.8.08.

(P.) 318/08. William Cochran Boyd. Locking for set screws. 13.8.08.

(P.) 319/08. Thomas Sneddon (1), John Sneddon (2). Improvements in rock and like crushing apparatus. 13.8.08.

(P.) 320/08. Arthur Edgar Dainton. Improvements in building slabs and construction thereof. 14.8.08.

(C.) 324/08. Pierre Germain. A process for separating precious metals and other metals which are combined with their gangue or enclosed therein, and apparatus therefor. 14.8.08.

(C.) 325/08. Walter Alson Hendryx. Method of recovering solutions. 14.8.08.

(C.) 326/08. Frank McKay Allan. Improvements in the manufacture of clinker for making Portland cement. 14.8.08.

(C.) 328/08. William Charles Stephens. Improvements in or connected with percussive rock drills. 14.8.08.

(C.) 329/08. William Charles Stephens. Improvements in or connected with rock drills. 14.8.08.

(C.) 332/08. William L. Imlay. Filter. 18.8.08.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

BLUME, E. E., *l/o* Ookiep; *Retain Journals.*

CLENNELL, J. E., *l/o* Gold Coast; c/o G. H. Clennell, Esq., 6, Great James Street, Bedford Row, London, W.C.

DRAKE, FRANCIS, *l/o* London; c/o American Institute of Mining Engineers, 29, West 39th Street, New York City.

GARDINER, C. T., *l/o* Johannesburg: Cinderella Deep, Ltd., P. O. Box 75, Boksburg.

HAMILTON, J. L., *l/o* Fordsburg; P. O. Box 678, Johannesburg.

HARDACH, E. E., *l/o* Reno Nevado: Witwatersrand Deep, Ltd., P. O. Box 5, Knights.

JOHNSON, TOM, *l/o* England; P. O. Box 370, Johannesburg.

LINTON, ROBT., *l/o* Mexico: 616, Bixel Street, Cal., U.S.A.

MACKINTOSH, H. G., *l/o* Scotland; Dunraven Hotel, Selukwe, Rhodesia.

MERRILL, C. W., *l/o* Alamedo; 143, Second Street, San Francisco, Cal., U.S.A.

MITCHELL, A., *l/o* East Rand; Ritanga Syndicate, Surprise Store, Selukwe, Rhodesia.

NICHOLLS, J. C., *l/o* Korea; New Modderfontein G. M. Co., Ltd., P. O. Box 25, Benoni.

OSTERLOH, E. A., *l/o* Krugersdorp; Simmer Deep, Ltd., P. O. Box 178, Germiston.

QUIN, S. C., *l/o*
P. O. Box 26, Maraisburg.

REECE, F. B., *to* P. O. Box 955, Lead, South Dakota, U.S.A.

ROBINSON, R. W., *l/o* Johannesburg; Penh[illegible]nga Proprietary Mines, Ltd., Umtali, Rhodesi[illegible]

ROSE, A. F., *l/o* Krugersdorp; Geduld Proprietary Mines, Ltd., P. O. Box 41, Springs.

TRUSCOTT, S. J., *to* 65, Aberdare Gardens, South Hampstead, London, N.W.

YEATMAN, POPE; *to* Room 1714, No. 165, Broadway, New York.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. IX. SEPTEMBER, 1908. No. 3.

Proceedings
AT
Ordinary General Meeting, September 19, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, September 19th, Mr. R. G. Bevington (President), in the chair. There were also present:—

39 Members: Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, K. L. Graham, C. B. Kingston, A. Richardson, C. B. Saner, H. A. White, Prof. J. Yates, A. Avent, H. D. Bell, E. Browning, J. S. Cellier, F. W. Cindel, A. A. Coaton, G. A. Darling, L. Evans, G. Goodwin, J. H. Harris, J. I'Ons, T. Johnson, J. A. Jones, G. A. Lawson, Hy. Lea, J. Lea, S. Morison, P. Morrisby, W. D. Morton, E. Pam, C. F. Parry, J. F. Pyles, W. H. Roe, C. E. Rusden, R. Stokes, W. Taylor, J. F. Walker, J. P. Ward, and J. Watson.

10 Associates: Messrs. J. Atkinson, J. Cronin, W. J. Dunnachie, N. Newland, W. S. V. Price, H. Rusden, W. A. C. Tayler, W. E. Thorpe, C. Toombs and W. Waters.

9 Visitors and Fred. Rowland, Secretary.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. W. E. Thorpe and W. Taylor were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows:—

Bennett, Randolph White, Witwatersrand G. M. Co., Ltd., P. O. Box 1, Knights. Cyanider.
Blewett, Edward, c/o Wilkins & Co., Chemists, Selukwe, Rhodesia. Chemist and Cyanide Manager.
Carr, W., Salisbury G. M. Co., Ltd., P. O. Box 1022, Johannesburg. Mine Manager.
Creed, John Percy, Village Main Reef G. M. Co., Ltd., 12, Eloff Street, Johannesburg. Cyanider.
Edwards, A. L., P. O. Box 74, Barberton. Assayer and Cyanider.
Kerr, Thos., Jumpers G. M. Co., Ltd., Cleveland. General Manager.
Strang, John, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Cyanider.

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting:

Allan, John Richard, Turf Mines, Ltd., Schuller Street, Forest Hill, Johannesburg. Miner.
Lindberg, Birger, Lebong Soelit, Ketaun, West Coast, Sumatra, Dutch East Indies. Mining Engineer.
Smith, Sydney A., Turf Mines, Ltd., P. O. Box 5887, Johannesburg. Miner.
Wolvekamp, Hendrik, Lebong Soelit, Ketaun, West Coast Sumatra, Dutch East Indies. Mining Engineer.

And the following Student:

Lipschitz, Cecil, Transvaal University College, P. O. Box 1176, Johannesburg.

General Business.

The President: It is very pleasing to see that we have so many gentlemen coming forward as Members and Associates. It is very satisfactory that we are able to keep up our list of members, and replace any resignations. In fact, we are not only able to do this, but we are increasing our membership.

MINES TRIALS COMMITTEE.

The President: There is one correction I wish to make with regard to what I said on the evening of my inaugural address respecting the Mines Trials Committee. I then stated that I understood the Mines Trials Committee interested itself in everything connected with mining. I have lately been informed that it does not do so, but that its scope practically embraces everything from the collar of the shaft to the residue dumps and slimes dams. I am very sorry to find that the Committee does not interest itself in appliances connected with winning the rock, that is to say, rock drills and other similar appliances, as well as the reduction works plant. I certainly think that the Committee should have laid itself open to deal with everything connected with mining, both on the surface and underground. I must apologise to any gentlemen who have been misled, and I know there are some who have already placed matters connected with rock drills before this Committee only to be informed that it could not deal with them. I can only say that I was misinformed, and that I regret very much if these gentlemen have been put to any inconvenience.

NOTE ON A PROBLEM DURING SHAFT SINKING.

By Chas. B. Saner, M.I.M.M. (Member of Council).

Shaft sinking on the Rand has on the whole been more or less plain sailing, and there are few recorded cases of how great difficulties met with have been surmounted.

On a certain mine on the West Rand very soft stuff was cut through with great difficulty, but unfortunately there have been no published details as to how the job was accomplished: on the East Rand water in one or two instances was a great hindrance, but no paper has been written showing the method of overcoming this bugbear. This note is a short account of how a difficult and dangerous stage in vertical shaft sinking was passed through in the shaft of the Turf Mines, Ltd., without loss of life and without undue loss of time, and will, I hope, encourage others to give the Society the benefit of their experiences in other such cases.

Shaft sinking and development is to my mind one of the most important and least acknowledged branches of mining: because on the excellence or badness of the work depends the future economical working of the mine to a great extent. If the main thoroughfare is bad and poorly laid, the running costs must be high; be it rail-road or shaft-road, one is horizontal, the other is vertical, but both are for the same purpose. The efficiency or otherwise of the shaft will affect the maintenance costs throughout the life of the mine.

The Turf Mines shaft is being sunk by hand labour of three 8-hour shifts, 75 to 85 boys are employed in lashing and drilling, 35 to 45 holes of 30 in. to 40 in. in depth being blasted per shift. Hand labour is much speedier and less expensive than any other method at present utilised.

The shaft is a seven-compartment one with its long axis running N. and S., that is across the formation, the north or pump compartment being No. 1, and the south compartment being No. 7, the bearers being similarly numbered.

The reason for sinking the "deeper deep" vertical shafts of the Rand with the long axis across the formation is for greater strength; no long ledges of rock are exposed on the strike, as is often the case when the long axis is parallel with the strike of the formation, the excavated area measuring in this instance 48 ft. N. and S. and 8 ft. E. and W. that is only 8 ft. of formation dips into the shaft, and as stage winding is to be adopted in these shafts, it is not necessary to consider the question of the bend.

For some time the shaft in being sunk was cutting through series of almost vertical faults with approximately S.E. and N.W. strike and dip S.W., cutting diagonally across the shaft.

On referring to Fig. 1 (plan of shaft at 2,640 ft.), it will be seen how the country rock around the shaft was cut up.

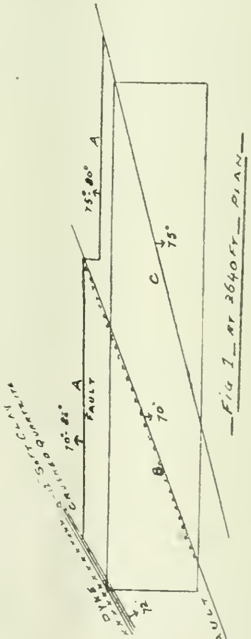


FIG 1 — AT 2640 FT — PLAN

Fault "A" was dipping steeply east 75° to 85°, with very distinct striation marks, and was cut through by Fault "B," which had about ½ in., of dyke matter on its line of cleavage. "C" was a fault plane crossing the strike of the formation in a diagonal direction.

The formation, although normally with a dip of about 34° S., was shattered and cut up badly around the dyke, faults and cleavage planes being thrown out in all directions and at all angles. The dyke on first being cut, was soft and shaley, with quartz seams running through; on the line of contact was 6 in. to 12 in. of pulverised clayey matter adjoining 12 in. to 36 in. of crushed and altered quartzite.

At 2,630 ft. a fault striking N. and S. dipping east appeared on the whole east side of the shaft. The rock was continually scaling off from this fault, and as the shaft was breaking wider and wider and a dyke face with a dip of 72° S.W. had appeared in the S.E. corner, it was decided to put in a set of bearers at 2,662 ft., hitches of 12 in. to 48 in. on the east side and 6 in. to 12 in. on the west side, and bearers 9 in. wide under the end pieces, and 7 in. wide under each divider by 27 in. to 36 in. deep, which were all firmly wedged in place.

On referring to Fig. 2, the reason for the great difference in length of hitches will be appreciated; the N. and S. hitches were 12 in. deep, being cut out of good solid ground, while the others were in "baulky" ground and so cut deeper for greater hold and security.

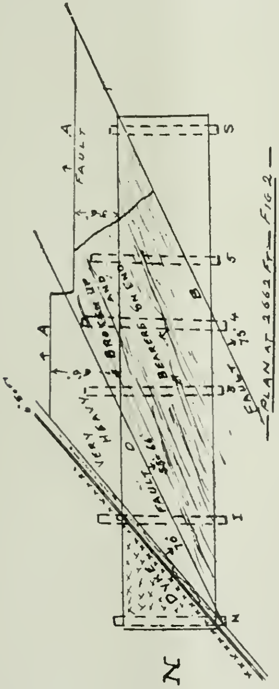


Fig. 2 gives a clear idea of the disturbances and the positions of the bearers. These bearers were wedged tightly from the west side, so as to resist any tendency of the east side to heave off to Fault "A" into the shaft.

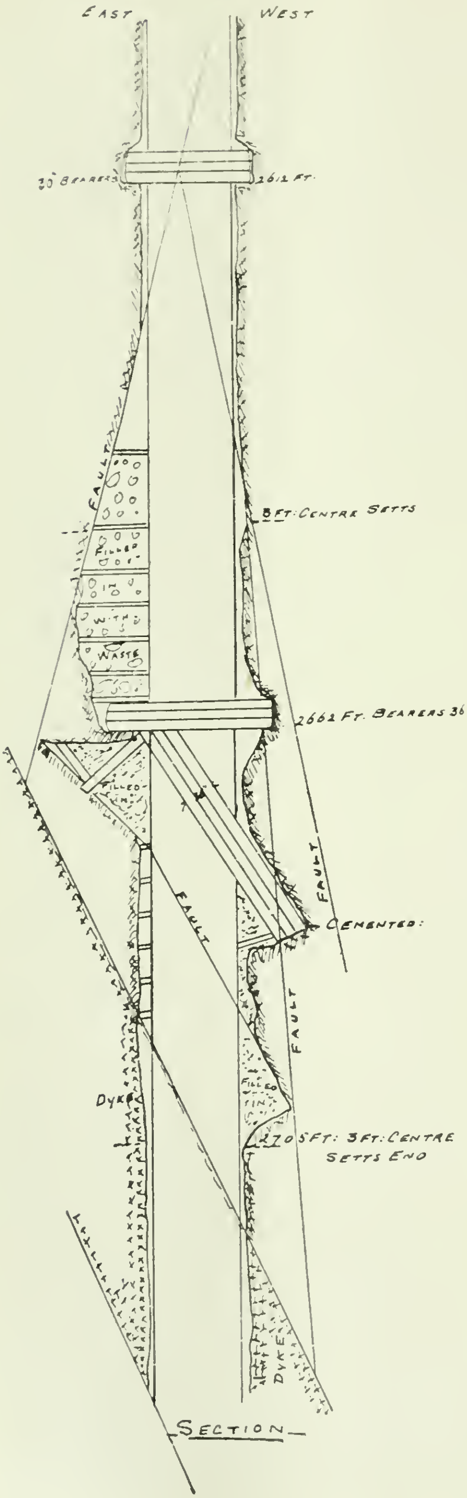
On the 2,662 ft. bearers a solid "bottom" was placed; that is, cross timbers from one bearer to another were firmly packed in, and all intervening spaces were closely filled with blocks, thus making a compact floor between sets and shaft side; then the sets were closely lagged with 3 in. to 4 in. Australian hardwood (stringy bark) planks and filled in behind with waste rock from the shaft bottom.

The method of filling in was as follows: The skip was loaded at the bottom in the usual way, and by signal slowly raised to the required position, where a white man was stationed. The clamping arms preventing the skip from tipping were then disengaged, a hook attached to a wall plate by a chain on the opposite side of the shaft to where the waste was to be tipped, was placed in a bracket on the bottom side of the skip, and then the signal to "lower slowly" given; the chain and hook took the weight and thus overturned the skip; the lip of the skip landing on a wall plate and the contents falling behind the lagging between fault and timber, thus making a solid side, and without a fragment of rock falling upon the 75 to 85 boys working below.

Previous to the above hitches being cut, it was decided to shorten two sets, which had been hung at 6 ft. centres to 3 ft. centres to catch the "heaving" ground more securely. The hanging bolts in the top set were first loosened, then the studdles were removed and brought to surface in the skips. Chains were lashed round the east and west wall plates of the lower set in Nos. 2 and 6 compartments and attached to the north skip of one engine and the south skip of the other engine by means of shackles below the skips; by signals given simultaneously to the driver of each engine, the whole was raised together; that is wall plates, end pieces and dividers in one lift; when raised to the required position, the long hanging bolts were taken out and the short ones put in and the set hung. The same was done with the lower set. The great secret was to get the two engines to work exactly together; a ticklish job and accomplished successfully and without mishap: this saved a great deal of time, as it obviated raising all the pieces of timber separately to surface and sending them down again.

As soon as all was secure and safe, sinking was resumed; but we had only got a few feet below our 2,662 ft. (or No. 24 set) of bearers, when the greater part of the east side below the bearers slid away into the shaft, fortunately immediately after the 3 p.m. blast, along the Fault "D," undercutting bearers No. 3, 4, 5. (See vertical section.) On careful examination the east side appeared so heavy and dangerous, that all sinking was again stopped; and to make assurance doubly sure, it was decided to come up the shaft above the region of these disturbing faults and put in another heavy set of bearers.

So at 2,612 ft. this was done. A complete set of eight bearers 30 in. deep in good solid hitches was put in; we usually place bearers under each



end piece and every alternate divider that is five in all, and about 100 ft. to 120 ft. apart. Moiling and cutting hitches in solid quartzitic rock is a slow, expensive, and expert job, especially with timber, which must not be damaged, close around, and there is always great difficulty in getting natives to moil out the hitches, because of the confined space and awkward positions; the method of cutting hitches is as follows: Having lined out the exact position carefully from the dividers above, diagonal or stripping holes are drilled about 6 in. to 12 in. deep, these are shaken with one-half to one stick of gelatine; then the hitch is squared by drilling holes 6 in. to 9 in. deep, as close together as possible, and the intervening rock cut out with chisels or popped with explosives, and finally the bottom is moiled or chipped square; a "lead" is cut away at one end to bring in the bearer timber; this lead may be either vertical or diagonal, it is usually the latter.

A "lead" is vertical when there are no dividers close to the hitch, so that the bearer piece can be dropped into the hitch. But usually diagonal so as to get the bearer in under the divider, and when once in the hitch, no side blow can dislodge the bearer. A "lead" is a recessed way cut out of the solid rock to give room for a bearer to be brought into a hitch.

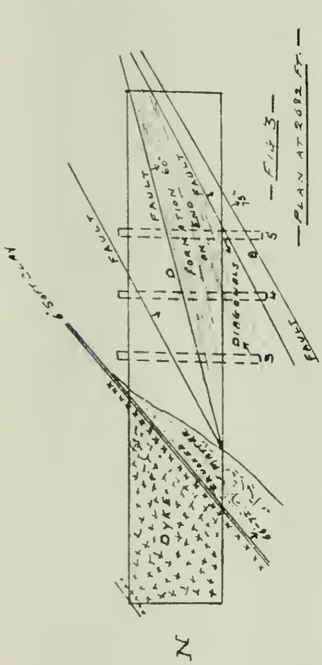
A "bearer" is a foundation girder and should always be several inches longer than the shaft is wide; the object of the bearer is to support the superincumbent weight and transfer this weight through the hitches on to the solid rock. A "hitch" is a step cut out of the solid rock to receive the end of a bearer.

The great thing is to get the bearer piece as good a fit as possible in the hitch, so that it should have no chance of being pulled out, in case of skip accident; a bearer piece should never be able to give way, but must rather break before being able to be dislodged from its hitches. Finally, the bearers 30 in. deep made up of three 10. in. pieces of pitch pine, were firmly placed in solid hitches under every divider and with the hanging bolts $1\frac{1}{8}$ in. diameter, and 12 in each set in place and each set well blocked, sinking was resumed, but with great caution and very lightly charged holes.

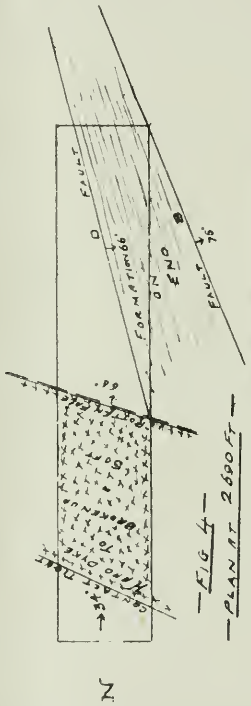
The undercut, eastern side, below the 2,662 ft. bearers was temporarily supported bv three legs 14 in. by 16 in. on a 12 in. by 14 i[illegible]ll 15 ft. long, well spragged. At 2,682 ft. [illegible]e deep hitches 4 ft. to 5 ft. were cut in the west side of the shaft and diagonal bearers were firmly placed with their heads under Nos. 3, 4, 5 bearers at 2,662 ft. and the toes against the fault on the west side and cemented in. These three diagonal bearers were two 36 in. deep and one 24 in. deep, consisting of 12 in. pieces of pitch pine, a great thickness of timber, and may be too strong, but it is better to be sound in time than to be sorry when too late.

These diagonals took a long time to get into position, as a neat fit was essential, and they had to clear the steel guides. They are at an angle of 57° upwards to the east, so supporting the heavy eastern wall of the shaft as well as the ends of the horizontal bearers; when these diagonals were in place the temporary legs were removed.

The ground between Faults D and B (see Figs. 3—4) was all on end and greatly disturbed and broken up, and as sinking was continued the trouble was transferred to the west side of the shaft, but fortunately only the southern half.


— Fig 3 —

— Plan at 2692 ft. —


— Fig 4 —

— Plan at 2690 Ft —

The sets at 3 ft. centres were continued to 2,705 ft., and, where possible, close lagged with 3 in. stringy bark and extra blocking behind every divider, firmly wedged in place and afterwards filled in with waste and ashes.

The timber was kept within 6 ft. of the bottom during this sinking, hence very light holes and careful blasting was necessary. The method of procedure was as follows: The north and south ends of the shaft were cut down with light holes so as to get as far away from the hitches on the west side as possible before proper blasting was done; then as the west side was so disturbed, a channel was cut along the east side and the ground stoped away from the west side, with light holes looking east.

By this means the shaft was slowly sunk until regular formation through the dyke was reached. No one unless he has actually worked with them can imagine the difficulty of handling heavy and large timbers in such confined spaces; when the sets were being kept close to the bottom, it was a difficult and slow job to hang the wall plates, each half plate being about 23 ft. long. They were dropped vertically to the bottom, suspended by a wire rope in pairs below the skip, and the ends dragged north and south till the plates were horizontal on the bottom, they were then detached from the skip and hauled up by block and tackle horizontally into position and suspended on the hanging bolts.

I am a firm believer in good solid hitches and well placed bearers, for on the soundness of these the whole shaft depends, and I deprecate the practice I have sometimes seen adopted of scamping the hitches, and only wedging the so-called bearers in place. Good bearers and solid hitches are a firm foundation for the shaft.

I may mention that before the bearers were wedged in place, four heavy plumbobs were suspended from the surface to ensure that the shaft timbers were in their correct vertical position before final blocking. Two bobs were lowered in No. 2 compartment, 10 ft. 6 in. from north corners on the east and west sides and 6 in. from the wall plates. Two other bobs were lowered in No. 6 compartment, a similar distance from the wall plates. Saw cuts had been placed in the set to be plumbed at 2,662 ft.

The lines were lowered from surface from reels, light weights being attached for this purpose, then the heavy bobs were put on, and while coming to rest the line was examined carefully from bottom to surface to make sure all was clear, the skip ascending slowly. After an hour, to allow the bobs to settle, careful measurements were taken and the set was found to be $\frac{1}{16}$ in. too much to the north and $\frac{1}{4}$ in. too much to the east.

The work now completed is a safe and good one and reflects great credit upon Mr. Coombe and the timbermen for the efficient and speedy manner in which it was done. No man shirked his job, though for several days and nights double shifts were worked, and the danger of working in a confined space below unknown masses of heavy ground was great, and if it had started to move, all would have been crushed like rats—that is the luck of the miner!

The President: Mr. Saner is very much to be congratulated on the lucid way in which he has placed this matter before us. This is one of those problems in mining which are very difficult to contend with and very interesting when the work is safely accomplished. Mr. Saner, and also Mr. Coombe, are to be congratulated on having safely performed a hazardous operation, and I trust they will have no further trouble in that quarter.

ZINC DUST PRECIPITATION AT CERRO PRIETO.

By Robert Linton (Associate).

The Cerro Prieto plant of the Black Mountain Mining Company, situated near Magdalena, Sonora, Mexico, is the only plant in Mexico operating on a large scale, and one of the few plants on the Continent, which is successfully using zinc dust for precipitation. The following account, which is confined to the matter of precipitation alone, and is not intended to cover any other features of the cyanide practice at that plant, may therefore be of interest to the members of the Society.

The ore is crushed in weak potassium cyanide solution in a 120-stamp mill, and the coarse sands reground in tube mills; the pulp is treated in a sands-slimes plant, the sands by percolation, and the slimes by decantation and filtering in vacuum filters. Weak solutions are used, 0·15% to 0·2% for leaching the sands, 0·075% for crushing and for slimes treatment. The gold dissolves rapidly, over 60% of the total contents being extracted in the mill. The gold solutions carry generally 2 to 3 dwt. gold and about four times as much silver; they contain ordinarily from 0·07% to 0·1% KCy and are of about 0·2% alkalinity.

The solutions drained from the sands and decanted from the slimes flow to the gold sump. A clarifying filter is placed in the top of the tank to remove any slimes in suspension. The filter is of the ordinary type used for filter bottoms, a wooden grating covered with cocoa matting and burlap, on which is spread 2 in. of coarse gravel and 6 in. of sand. The occasional cleaning required is quickly and easily done by scraping a thin layer of sand from the top. As the filter is in the top of the tank, this can be done without emptying the tank, and only requiring the flow of solution to be shut off for a few minutes. The solution filters through by gravity, is drawn off as convenient and pumped to the precipitation tank on the hill above the plant. The latter has a storage capacity of 450 tons, and has in the bottom a clarifying sand filter similar to that just described.

The gold solution is drawn off from the bottom of the precipitation tank through a 6 in. pipe connected to the zinc press. Zinc dust in the form of an emulsion is added at the point where the solution leaves the tank, the emulsion being prepared as follows:—A small quantity of solution is lifted by compressed air through a 1-in. pipe and discharged into a cone at the top of the tank. Zinc dust is fed into this cone at regular intervals and thoroughly mixed with the solution by agitation with compressed air introduced through a $\frac{3}{4}$ in. pipe reaching nearly to the bottom of the cone. The emulsion overflows into a 3-in. pipe leading into the 6 in. outlet pipe, is added to the outflowing gold solution, and becomes thoroughly mixed with it in flowing down to the zinc press. This apparatus maintains very satisfactorily the essential regularity of zinc dust feed.

The zinc press is of the Dehne type, 35 in. square on outside of frames, having 24 leaves with 4 in. frames intervening, the total filtering surface being 367 sq. ft. Cotton sheeting 72 in. wide, made by the Pepperell Mills, Bradford, Maine, is used for filter cloths. Two cloths are used over the leaves. Each time the press is cleaned new cloths are put on next to the leaves, the inner cloths put on over them, and the outer cloths burned The press stands 31 ft. below the bottom of the precipitation tank, affording a head of from 31 to 47 ft., depending upon the height of solution in the tank. No pump is used in connection with the press, the flow being maintained by gravity alone. The pressure corresponding to head would vary with the height of solution from 13·4 to 20·4 lb. The working pressure of the press, or the back pressure of the solution flowing through it, rises from zero, when the press is started, to a maximum of 14 lb. before the break.

After a break of the press, precipitation is started by feeding an excess of zinc dust into the feed cone, then opening the valve to the press, allowing the mixture to flow through slowly, in order to form a thin film of zinc dust on every filter cloth. The high rate of zinc dust feed is kept up for six hours, the quantity of solution fed being meanwhile gradually increased. At the end of six hours, the rate of feeding zinc dust is cut down and gradually lowered until the press is broken again. Before starting the press the gold solution is always titrated for KCy, and when necessary enough fresh cyanide added to insure the immediate setting up of precipitation. After precipitation is once started, comparatively weak solutions (as low as 0·07% KCy) are precipitated without difficulty. The precipitation is very rapid and complete. As much as 1,200 tons of solution has been passed through the press in 24 hours, with an average value left in the effluent solution of less than 1 gr. gold and a mere trace of silver. This means that the solution is flowing down through the pipe line at a rate of 135 ft. per minute, that the entire volume of the press is being displaced every three minutes, and that the total time that the gold solution is in contact with the zinc dust is less than five minutes. The rate of flow corresponds to 3·2 tons of solution per square foot of filter surface per 24 hours.

Lead acetate is added in suitable proportion to the zinc dust, which assists the precipitation. The silver contents of the solution doubtless facilitate as well the rapidity of the precipitation.

With accumulation of precipitate in the press the rate of flow through it gradually decreases with a corresponding increase of pressure to a minimum of about 700 tons per 24 hours, the average being usually about 900 tons per 24 hours for the entire time from starting the press until it is started again, including the time required for cleaning. This tonnage, it will be understood, is handled from the one precipitation tank and through the one zinc press.

The consumption of zinc dust is 0·4 lb. per oz. of gold and silver precipitated. Calculating this on the actual metallic zinc available in the zinc dust, the consumption is 5½ parts of zinc to one of gold and silver. The press is broken and cleaned four to six times per month. The precipitate is dried, briquetted with borax and litharge, and melted and refined in an English cupelling furnace.

Without going into an extended discussion of the respective merits of zinc dust and zinc shavings for precipitation, a few of the advantages and disadvantages of each may be noted, together with a few of the essential points to be noted in using the former.

Success in zinc dust precipitation requires careful supervision and a thorough working out of questions of feed of zinc to the solution, and of zinc and solution to the press. The aim is to obtain a granular product which, when it forms a coating on the cloths or settles to the bottom of the frames, shall offer the least obstruction to the flow of solution through it. If the product is very fine, or contains slime carried in from unclarified solutions, the coating on the cloths soon becomes dense and impervious to the solution flowing through the press, and the capacity of the press consequently diminished.

Gravity feed appears to offer an advantage over the use of a pump to force the solution through the press. The tendency of a pump is to break up the product and produce the fineness that it is aimed to avoid, while when the zinc and solution flow quietly to the press, as is the case in using the gravity feed, the condition is the most favourable for forming the granular product, and the fullest capacity of the press obtained. To the same end a thorough clarification of the solution prior to precipitation is necessary so as to avoid any clogging of the filter through slime settling in it.

The refining of the zinc dust precipitate involves more labour and expense than the refining of the zinc shavings product. The latter is usually melted by the cyanide foreman, whereas melting the zinc dust product practically means the addition of a smelting department, on a small scale it is true, to the operations. More flux is used, more fuel, and more labour is required. The additional cost has proved a serious factor in many cases, and has even in some cases led to the abandonment of zinc dust precipitation altogether. On the other hand, there is the saving of all the trouble and expense connected with the zinc boxes ; and if the refining is done efficiently and the precipitation effected with a minimum of zinc dust, the cost of refining may be practically balanced in the saving made by having no zinc boxes to look after. To this must be added a further advantage, and the most important one derived from zinc dust precipitation, the increased security against theft.

Zinc dust used for precipitation should be regularly tested for total contents of metallic zinc, zinc oxide, and lead. It is frequently very impure, and unless the operation is regularly controlled by such tests there is likelihood of incomplete precipitation on the one hand or excessive consumption of zinc on the other. Too much zinc is not only a waste in itself, but materially increases the refining cost. The limited supply available makes it often difficult to obtain a satisfactory quality, and has been a hindrance to its more general use. Any considerable increase in the consumption of zinc dust could easily create a demand which the present productive capacity could not meet, but the eventual result would be the manufacture of zinc dust as a separate commodity instead of an undesirable bye-product of zinc smelting, and the making available of a more reliable supply, both as to quantity and quality, than the market now affords.

The President: We have to thank Mr. Linton very much for a most interesting paper, and I am very pleased to see that we are getting contributions from such distant parts as Mexico. There is no doubt that in these times Mexico is waking up with regard to metallurgical processes, as you will have noted from the able paper presented to the Society by Mr. Caldecott on the treatment of silver ores in Mexico. From all that we hear and from the correspondence between Mr. Caldecott and various gentlemen in Mexico, it is evident that a very great interest is being taken in metallurgical work there, and I feel gratified that we have this paper from Mr. Linton, which I hope will be the forerunner of many more from that part of the world.

DISCUSSION.

Mr Walford R. Dowling (*Member of Council*): I have read the author's paper with considerable interest, and only regret that he did not go into the subject more fully. Although the paper only deals with precipitation, he mentions the cyanide strengths used for dissolving as 0·15% to 0·2% for sands and 0·075% for crushing and slime treatment. I would like to ask whether these strengths are necessary for dissolving the gold and silver, or are they maintained so high for the purpose of precipitation? I like his method of bringing about the mixture and contact of solution and zinc dust, but am not clear as to how the zinc dust is fed into the mixing cone. Is some kind of mechanical feeder employed, or is this done by hand?

With reference to the sheeting used for filter cloths, apparently the outer sheeting is burnt at each opening of the press. At that rate, with four openings per month four complete sets of sheeting are burnt per month. In this country the item would amount to something in cost.

I can appreciate the point of aiming at a granular product to ease filter pressing, but do not see any mention of how this result is obtained other than gravity feed of solution. I have been trying to find out for a number of years what are the conditions which give an easy pressing of acid treated gold slimes, without much success. Gravity feed is probably better than pump feed, but in any case, if the slime is granular, the pressing is easy. My difficulty is how to keep the slimes always granular.

With reference to the notes on smelting, the author does not say why it is necessary to treat the zinc dust precipitate differently to zinc shaving precipitate, and why it involves greater expense. I understand that zinc dust in America is not more costly than zinc shavings, and where about the same quantity of either per ton is required the case improves for zinc dust. In South Africa zinc dust is still more expensive than zinc shavings, and would have to become much cheaper before the method could be considered.

I am afraid my contribution to the discussion consists mostly of questions, but I hope the author will be able to enlighten us on the points raised.

Mr. McArthur Johnston (*Vice-President*): I should like to put two questions before the Society to-night, so that Mr. Linton may have the opportunity of replying as early as possible. He does not definitely state here whether or not amalgamation takes place in the mill. This point might, I think, be made a little more clear. Then he says the silver contained in the solution doubtless facilitated the rapidity of the precipitation. Can Mr. Linton give us any figures showing the relation existing between the amounts of gold and silver precipitated by the zinc, and to what extent the quantity of silver present facilitates precipitation of the gold?

Mr. H. A. White (*Member of Council*): I notice that no mention of acid treatment of the zinc gold slimes is made and would suggest that this should considerably reduce the trouble and expense of the clean-ups.

The President: I think that point is an important one. We will transmit these queries to Mr. Linton, and in due course get his reply.

Mr. J. A. Jones (*Member*): I would like to know how the screen assay is obtained there. It will be interesting to us on the Rand to know how they get it in other parts of the world. Also how theoretical extractions and actual recoveries compare, and are the copper plates, if any, attacked very much by cyanide?

The President: Mr. A. Richardson has kindly undertaken to read the following paper:

SMALL MINES OF RHODESIA.

By B. I. Collings (*Member*).

In bringing a few notes on this subject before the Society, it is to be understood that no very new or very perfect methods of mining or metallurgy will be dealt with, but rather an attempt will be made to describe the nature of the ore deposits, the economic and the industrial peculiarities of the country, and their effect on the methods of mining and milling generally adopted. The subject may best be dealt with under four separate headings :—

1 geological, 2 economic, 3 mining, 4 metallurgical.

Geological.—The gold mines of the country are, as far as I am aware, all located in a geological horizon corresponding to that known as the Swaziland system in the Transvaal. This syst is recognised as the oldest series of rocks in Sou Africa, and it is in it that the gold mines of the Barberton, Klein Letaba, and Murchison districts of the Transvaal are situated. In Rhodesia it is represented by schists, banded ironstones, quartzites, conglomerates, and occasional slates; the conglomerate beds however, seldom bear any similarity to the banket beds of the Rand.

Most of the mines in Rhodesia work quartz reefs, which may be of any colour or appearance, but gold is occasionally met with in other matrices; in conglomerate as at the Eldorado and the Riverlea, in schist as at the Giant, and in altered granulite as at the Ayrshire; in fact, any rock in the gold belts of Rhodesia, if mineralised, is worth testing for gold. There is very seldom any definite formation in which one may more particularly expect to find gold, but it appears as a rough general rule that the payable mines are to be found either in schist near the contact of one or other of the big granite masses of the country, or close to some minor intrusion of an acid igneous rock. These granite masses, which are probably of similar age to the old grey granite of the Transvaal, are intrusive into the rocks of the Swaziland system, and though they are, generally speaking, very poorly mineralised themselves, it appears highly probable that they played an important part in the genesis of the ore bodies found in the metamorphic rocks. Contacts of all kinds are promising places to prospect, and these may often be recognised, even where there is a heavy overburden, by a change in the character of the bush, trees growing more readily on some soils than on others. There has been practically no prospecting for virgin reefs, nearly all the work having been confined to opening up ancient workings, of which there must be many thousands in the country.

The reefs are more or less lenticular and vary in width from a few inches up to 20 ft. or more. The distribution of the gold in them seems to follow no rule whatever, and base metals are usually present, pyrites being the most common, but galena, chalcopyrite, mispickel, and stibnite, are occasionally met with, although seldom more than two of these minerals are present in the same ore. A curious feature to be observed in some of the quartz reefs carrying gold at surface, is the disappearance of all values at depths of 30 or 40 ft., and, in others, the disappearance of the gold at water level; these reefs cannot all be regarded as gash veins, as they often continue in depth although barren. It is possible that the gold in them may have been precipitated by organic agencies, which would be effective only close to the surface. They often show a thin film of visible gold in the cracks of the quartz, which is very deceptive, as it pans disappointingly; it is known locally as paint gold.

It is not to be understood that a large proportion of the reefs lose their gold within 30 or 40 ft. of the surface, but it greatly increases the value of a prospect when it has been proved below water level. It must be admitted, however, that even where the gold holds below water level, a falling off in values with depth is by no means an uncommon phenomenon, and some mines that have been highly payable at surface, have been comparatively poor at depths of 700 or 800 ft.: but it is impossible to predict whether in any particular case an impoverishment may take place, as many of the older mines show as good values as ever in depth, while most of the small workers' mines are not yet deep enough to enable one to express an opinion.

It is idle in the vast majority of cases to search for the extension of a reef for any great distance.

Few mines in the country possess a pay shoot of greater length than 600 ft. or 700 ft., and in most instances the values only hold over much shorter strikes than this, a pay shoot of 200 or 300 ft. being considered quite a sound proposition for the small man. Payable mines are quite frequently found in groups within a mile or two of one another, but even in these cases the individual mines often differ greatly in the character of reef and mode of occurrence of gold, both free milling and refractory ores being encountered in the same belt of country.

In prospecting for virgin reefs, it must always be borne in mind that the ancients have been over the ground previously, and it is unreasonable to hope that any conspicuous outcrop carrying free gold will have been left untouched. Virgin reefs of value are very difficult to find, and many of those discovered have only been struck by accident. In examining an ancient working on which no present-day work has been done, the absence of any quartz on the dump is not a bad sign, but rather the reverse, as it often means that the reef was all of it sufficiently rich for the ancients to take away and crush. Ancient workings, however big, are not necessarily a sign of high gold values, and it seems probable that even in those far-off days there were "wild-cats."

The quartz reefs now being worked are seldom of any great width, a stoping width of 3 or 4 ft. being usual. The width in some mines is fairly consistent throughout the pay shoot, but more often the ore occurs in a series of lenses. When the ore body consists of ironstone, schist, or conglomerate, the widths are often very great, but the ore is of correspondingly low grade. Very few small workers have as yet attempted to tackle those big impregnations, and where they have been worked, only the richer portions have been exploited. There is little doubt, however, that the ultimate future of the country, from a mining point of view, rests with these big low grade ore bodies, and it is probable that as the small workers extend their operations, this type of mine will be more largely worked. In the gold belts the country is full of reefs, of which probably not more than one in fifty could be pronounced payable, some being too narrow, and others too poor. At the same time, a series of unpayable reefs will sometimes through weathering and disintegration, particularly where rich stringers are present, give rise to a payable rubble area. Although seldom of high grade, these rubble deposits are very cheaply worked, and are of great value when taken in conjunction with reef mining proper, as a stop-gap when development is behindhand, a state of things which, I regret to say, is the rule rather than the exception in the small mines. These rubble deposits often extend over many acres, but are usually payable only in patches; the depth varies from 6 in. up to 6 ft. or more. A certain amount of material classed as rubble consists of the sorting heaps of the ancients. The sampling of rubble is a very laborious operation; the area should be well trenched and big samples taken. Many expensive mistakes have been made in the estimation of rubble, and great caution is recommended before tackling a proposition of this sort.

To those accustomed to the comparatively even values of the Main Reef series of the Rand, an example or two will be given, showing the necessity of taking as little as possible for granted in quartz mining.

Case I.—Two large ancient workings were situated, in a line of smaller ancient workings, about 200 ft. apart from one another. Shafts were sunk in each of these to a depth of 70 to 80 ft., and excellent values were obtained. A 200 ft. shoot of good ore was assumed. On driving to connect the two shafts the values fell off at once, and further examination showed that the two shafts had been sunk in chimneys of ore caused by the intersections of two somewhat insignificant looking stringers.

Case II.—A shaft was sunk on a line of ancient workings, and, having bottomed the old workings, was continued for a further depth of 40 ft. on the reef. It was then decided to drive in order to determine the length of the payable shoot before striking the water. The drive in one direction showed abnormally high results, many ounces to the ton, and it was assumed that there were 40 ft. of backs of this grade ore. On starting to prepare to stope, the old workings were broken into within 18 in., and the assumed backs were found to be almost totally non-existent. It is true that neither of these cases should have proved a trap for a careful man, but there is a strong temptation for the small man, to whom money is scarce and time is valuable, to rush at conclusions and take chances that under other circumstances he would not feel inclined to do. At the same time, if a man is to do any good in a quartz mining country he must take risks. The man who takes chances and makes the fewest mistakes does best, and the man who waits till he is absolutely sure before acquiring any interest, will wait a very long time and often acquire nothing.

Economic Conditions.—The rate of wages paid in Rhodesia to whites has been higher than that obtaining in the Transvaal, and although the larger mines are now making a general move to lower these rates, it is probable they will still continue to be above those paid on the Rand. Miners and mechanics have received 25s. a shift in the past, and although the company owned mines have recently cut these rates, the small workers will probably continue to pay the old rates, as the hours worked are so much longer, and Sunday work is pretty frequent.

Native labour is cheaper than on the Rand, but probably not so efficient, while there is invariably a great shortage during the summer months. Surface labour is paid from 10s. to to 25s. a month, while underground boys receive from 15s. to 50s. a month, but rates vary considerably in different districts. Police boys, boss boys, and other special boys, receive 80s. a month or more. It is customary to start a boy at the lower wage, and gradually raise him according to his abilities. Boys are fed and housed by the mines as in the Transvaal. Many of the boys employed are recruited by the Rhodesian Native Labour Bureau, a semi-official concern similar in character to the Witwatersrand Native Labour Association, but the native labour position in Rhodesia is by no means a satisfactory one. Most of the boys supplied by the Bureau come from North-Eastern and North-Western Rhodesia, while most of the Southern Rhodesia boys come voluntarily to work. The high railway rates make machinery and stores expensive, and competition amongst the merchants is not carried to the same extremes as in the Transvaal.

Practically all the small mines use wood fuel; some of the bigger mines which have been working for many years, having cut all the wood in their vicinity, find it cheaper to use coal. The only producing coal mine is Wankies, situated on the Bulawayo-Victoria Falls Railway some 200 miles from the nearest gold mine. There are other well-known coal deposits much nearer the gold belts, but the Chartered Company is not in favour of their being opened up, as it is interested in the Wankies mines. In some parts of the country no charge is levied on the mines for cutting wood, but there is considerable difference of opinion between the farming and mining communities as to both wood and water rights. The position is rather complicated, as farms are held under two or three different forms of title, but in

Mashonaland, in almost every case, the miner has full wood and water rights without payment. Water is obtained either from rivers or from the mine, dams being very seldom seen.

In the Penhalonga district near Umtali, everything is driven by water power, but this is quite the exception. With regard to transport, most of the small workers ride their own cord wood and mining timber, which is cut in the vicinity of the mine, using oxen or donkeys for the purpose. At the stations and principal sidings transport is undertaken by forwarding agents and storekeepers. During the rainy season the transport of heavy machinery to any great distance is rendered almost impossible, as the roads are merely tracks cut through the bush, with frequent vleis where the wagons sink up to the axles in mud.

The climate is not unhealthy, and women and children are to be seen in every district. Good health is largely a matter of diet, and fresh meat and vegetables are much more easily obtained than they were ten years ago. One never hears of men compelled to leave the country on account of fever. Lions very seldom give trouble, but transport riders occasionally lose donkeys or oxen through them. White ants are troublesome, but there are various preparations, such as carbolineum, solignum, etc., which keep them away. Tar is also effective in preserving wood from white ants.

In the Rhodesian law, provided the outcrop of the reef is pegged, the claim holder can follow the reef on the dip as far as he pleases. There is therefore no such thing in Rhodesia as a deep level claim. Practically the whole country is open for pegging, and the holders of the surface rights have no claim at all on the minerals. In order to be able to peg ground, it is necessary to obtain a prospecting licence. Each prospecting licence costs £1, and enables the owner to peg out a block of 10 claims, measuring in all 1,500 by 600 ft. An individual can obtain as many prospecting licences as he pleases, on paying for them. The first step in pegging ground, after discovering one's reef, is to post a discovery notice. This protects the ground for a radius of 1,000 ft. for 30 days. Before the expiration of the 30 days, however, it is necessary to peg out the actual area required, and then apply at the nearest mining office for registration, giving a sketch and describing the position of the ground. This registration costs a further £1 for a block of 10 claims. Within six months of posting the discovery notice, it is necessary to do 30 ft. of development, or in default, obtain an inspection certificate at the cost of £5. During the next six months a further 30 ft. of work must be done, or in default an inspection certificate at the cost of £15 must be obtained. After the first year 60 ft. of work must be done annually, and inspection certificates are only granted on work done and cannot be obtained by payment. There are no claim licences to be paid as in the Transvaal. In the case of a number of contiguous blocks registered in the same name, the work can all be concentrated on one block. Many of the claims pegged years ago and held by companies are held under more easy terms than the above; inspection certificates being obtainable by payment, and work done on one block of claims is allowed to protect other blocks, whether contiguous or not. Blocks of claims containing ancient workings on which no work has been done, can be bought at from £50 to £250 or more, but ancient workings are by no means a guarantee of payability. Where properties are worked on a share or tribute basis, it is almost invariably arranged on a percentage royalty on the gold won. This is the only satisfactory way, and the one that least lends itself to any possibilities of dispute. The usual royalty asked is in the neighbourhood of 10%.

As to what constitutes payability, this depends very greatly on reef conditions, locality, etc., and these vary very widely. A 2 dwt. recovery on rubble may under favourable conditions be payable, while a 10-dwt. recovery on a reef where conditions are unfavourable may barely pay expenses. Under average economic conditions, a 10-stamp mill with a cyanide plant working a 3 ft. reef should work at from 15s. to 20s. per ton milled.

The capital required to start a small mine depends largely on the class of machinery put up, but I will give a rough estimate of the cost of three types of plant very often seen. Tremain steam stamp outfit with Dodge crusher, boiler, and pump, etc., price f.o.r. Salisbury, £720. Capacity, 300 tons per month. Five-stamp Californian mill, 1,050 lb. falling weight, with engine, crusher, boiler, pump, etc., price f.o.r. Salisbury, £1,450. Capacity, 750 tons per month. Four foot Chilian mill, with engine, boiler, crusher, pump, etc., price f.o.r. Salisbury, £1,400. Capacity, 800 tons per month. These figures are very rough, and the capacity depends naturally on the kind of ore crushed. A galvanised iron cyanide sands plant to treat 600 tons a month will cost erected between £400 and £500. To erect a 5-stamp mill and cyanide plant and develop a mine to keep it adequately supplied, cannot be done for less than £3,000, and this sum is usually considerably exceeded. Huts, tools, camp equipment, rails, piping, livestock for transport, and similar items, all run away with money, but fortunately credit is more easily obtained than in the Transvaal.

Mining Conditions.—These are, as far as the small mines are concerned, exceedingly simple. Shafts sunk on the reef are the rule. Sets made of round native timber poles are used, the favourite wood for this being "mopani," an exceedingly hard wood and difficult to work, but it is one of the very few that resist the attacks of borers. Most native woods become quite rotten within a year or so from these pests. The shafts of the small mines have generally only one hauling way, a single drum hoist with a ½ ton skip being sufficient to keep the usual small mine going. Native timber props are used in the stopes, but owing to shortage of development too few pillars are often left. Headgears are made of native timber or sometimes of 9 in. by 3 in. deals, and are seldom over 25 ft. high.

In most of the mines the reefs dip pretty steeply, and it is not often one strikes a quartz reef dipping at less than 50°. As a rule, the gold is free, and the values are followed by the pan, an assay plant on a small mine being very unusual. Drilling is all done by hand, compressed air being beyond the means of the small worker, and in most cases they are working in oxidised or semi-oxidised ore, where hand-drilling is in every way more economical. The character of the hanging wall varies in every mine, while the distance apart for levels depends more upon the fancy and resources of the man working the mine than on strictly mining principles; it is seldom more than 100 ft. Gelignite is the explosive generally used, the price of which is 42s. 9d. a case at the nearest station. Gelatine is 58s. 9d.

Nothing is known concerning the ancient workings, but it appears probable that some of them are over 200 years old. The method of mining was by fire and water, and charred pieces of wood are occasionally found in the old stopes.

Metallurgical Conditions.—Under this heading I will deal with surface plant generally. The special requirements of the small worker are portability and cheapness in first cost and erection. First of all I may mention that sorting is rarely indulged in by the small worker, as where the reef is narrow the casing for a few inches on both sides usually carries values.

One of the striking features of the surface plants on the small mines is the number of types of crushing machinery employed. Tremain steam stamps, Huntingdon mills, Chilian mills, pneumatic stamps, and last, but not least, Californian stamps, are all to be met with. The advantages of the Tremain mill are portability and low first cost, its chief disadvantage being heavy steam consumption. Huntington mills are low in first cost and use little power, and in crushing soft clayey ores their output is large and cost of spares small; for hard ores they are very inefficient and expensive. Chilian mills are higher in first cost than Huntingtons, but for soft or moderately hard ores, treaing large tonnages, they are very satisfactory. Both the Huntington and Chilian mills, when treating the harder ores, are used to best advantage as secondary crushing machines. The pneumatic mill, it is claimed, costs less and has less weight, for a given capacity, than the Californian mill, and is probably well adapted for very hard ores. Two, three, and five stamp mills of the Californian type are frequently seen, and the old gravity mill is under many conditions very difficult to beat, particularly when subject to the rough usage machinery receives in the hands of some of the small workers, who are too apt to call in the aid of a 7-lb. hammer whenever things go wrong.

Frenier sand pumps are largely used for lifting tailings, and for heights up to 18 or 20 ft., they are admirable, requiring little power or attention. The Frenier spiral sand pump consists of a spiral ribbon of steel plate in form like a spiral clock spring. On each side is a steel disc, which is joined to the spiral by continuous air-tight joints, thus making a spiral tube of steel with a rectangular section of constant area throughout. It is mounted on a hollow horizontal shaft, which has an opening to connect with the spiral tube. There are no valves, but the water and sand are raised by virtue of a hydrostatic head in each turn of the spiral, a part of each turn being filled with water and the rest with air (the pump being partly immersed in water and partly in air). The pump works best at about 20 revolutions per minute.

The cyanide tanks on the small mines are invariably made of galvanised iron, and are generally placed directly on the ground, and are filled and emptied from the sides. For this reason it is better to have the units small, and 15 ft. is the largest diameter tank that can be conveniently worked in this manner. The cost of a 15 ft. by 4 ft. G.I. sands tank complete with filter frames and matting is about £40 to £45. The battery pulp is usually first run into a settling tank fitted with a slimes gate, and the sand thus collected is transferred to the treatment tanks; but in some cases it is run into a series of pits, where it settles and is afterwards dug out. Most small workers have very little idea what their percentage recovery is, as many cyanide plants are run without any assays ever being taken, but below the oxidised zone it is probably well below rather than above 80%, for very few small mines have slimes plants, as their first cost is so heavy. Blanket strakes, or canvas tables, are only occasionally employed, and their use combined with some small amalgamating pan might probably be extended with advantage.

Duplex steam pumps are generally used for cyanide solutions, as it often happens that there is no suitable line shaft from which to run a centrifugal pump. The usual type of boiler used is the locomotive, one of its advantages being that it requires no brick-work setting. Slide valve engines are almost invariably used, and where fuel is expensive, feedheaters are installed. Gas producer plants are spoken of very favourably, and will probably be largely used in the future. Electricity is seldom used, but might often be advantageously employed for working isolated pumping stations; most small workers, however, understand very little about it, and are therefore afraid of installing it. Water is a very important question on many mines, it being necessary to pump sometimes three or four miles. One often sees the mistake made of putting the mill by the water instead of carrying the water to the mill, but in many instances a three or four mile pipe line is altogether beyond the resources of the small man. When debating on the introduction of labour-saving appliances, local conditions must be considered, and in Rhodesia where labour is cheap and machinery expensive, and where moreover the life of a mine is such an uncertain factor, labour-saving appliances are not justified to the same extent as in most other fields. With limited resources, the deciding question is often, what *can* be done, rather than what *ought* to be done.

In Rhodesia one must regard almost every individual mine as a law unto itself; character of ore, character of walls, character of gold, all vary; in no single way does it appear possible to generalise on them. For instance, I will take a mine with which I am personally connected. It is called the Cam and is situated in the Hartley district. The ore consists of about 30% quartz and 70% chlorite schist, the walls being of schist. The recovery over the plates is at present less than 20%, while the slime charges are more than half as high again in value as the sand charges. The ore contains stibnite and mispickel, and shows at depth no gold in the pan, the reef width is about 6 or 7 ft., the footwall is defined, but the values gradually merge into the hanging. The stibnite occurs in patches and carries high gold values, which have up to the present proved irrecoverable. The bullion contains practically no silver and has been as high as 991 fine. At one place in the mine a cross-reef occurs and values are obtained over a width of 30 ft. or 40 ft. Within little over a mile of us is the Eiffel Blue mine, which works a quartz reef in what I am told is diorite. Here 80% of the gold is caught over the plates, and the slimes are not rich enough to treat. The ore contains no stibnite, while the bullion contains an appreciable amount of silver.

The above notes will, I hope, enable members of the Society to form some idea of the conditions affecting the small worker in Rhodesia.

Bibliography.—Hatch and Corstorphine: Geology of South Africa.

Carey: Gold Mining in Matabeleland. Institute of Mining and Metallurgy, 1901-1902.

F. P. Mennell: The Geology of Southern Rhodesia. The Mineral Wealth of Rhodesia. (Argus P. & P. Co., Bulawayo.)

J. H. Curle: Gold Mines of the World.

A. R. Sawyer: The Goldfields of Mashonaland.

Charles Hunter: A Cheap Form of Cyanide Plant.* Institute of Mining and Metallurgy, 1907-1908.

The President: The thanks of our Society are due to Mr. Collings for placing this paper before us. It is very interesting indeed and should call forth a good deal of discussion; Mr. Collings points out in one place that the ultimate future of the country rests on the big low grade ore bodies. I do not think that that only refers to Rhodesia. Reduced working costs and improved facilities in working generally, will allow us to develop our own large bodies of low grade ore and that will very materially affect the life of the mines of this country. I should like to make one remark on the subject of mining papers. It has been stated elsewhere that there is a great paucity of mining papers in Johannesburg. I cannot agree with the gentleman who made that statement. Here are we to-night with two such papers before us, and we have had several others recently, one of which is under discussion now. I refer to the paper by Mr. Tom Johnson who, I am sure, will give us a very able reply to-night. This contribution has given great food for discussion and reflection. I must say that I cannot agree that there is with us a paucity of mining papers, though it may be that in another place there is.

NEW TYPE OF NATIVE COMPOUND BUILDING OF ALL METALLIC CONSTRUCTION.

(*Read at April Meeting, 1908.*)

By C. B. Kingston, B.A., B.A.Sc. (Member of Council).

REPLY TO DISCUSSION.

Mr. C. B. Kingston: Replying to Mr. Schmitt's discussion of my short paper. The question of the merit of this type of building as a habitation for natives may perhaps be a matter of opinion, but as stated in my paper, the design received special commendation from a well-known authority on native housing. His opinion has been supported by our experience. The health of our boys has been extremely good since they moved into the new compound.

---

* See this *Journal*, vol. viii., Sept., 1907, pp. 90-91.

During the past winter, the rooms, which are heated with ordinary stoves, have been quite warm and comfortable. They will, doubtless, be warm on hot summer days, but so will any other building with a single roof of corrugated iron. The iron building has this advantage, that its walls do not radiate heat during the night as brick walls do, but cool quickly, after sundown, and remain cool during the hours when the boys are resting.

On the plans submitted, the windows are stated to be equivalent to 10% of the floor space.

Sloping windows, set high in the roof, receive the sun's rays more directly than vertical windows, and transmit the light directly into the heart of the room. This point received special consideration, and after discussion with the health authorities, it was decided that windows set as these are set would produce an excessive illumination if made with an actual area equal to 10% of the floor space.

Iron of 24 gauge was not used without first testing the protective coating after bending to the required curve, and although well aware that heavier iron is commonly used for curved roofs, I am satisfied that the 24 gauge iron is heavy enough for the building in question, while the riveted joints appear to be perfectly satisfactory.

Care is taken to prevent the boys using open fire boxes, so that the roof may not be exposed to dangerous fumes.

As to the prophecy that the life of the building will be shorter than expected, I am sorry that Mr. Schmitt is unable to take a more cheerful view, but now that the building is up, we can only wait and see what happens.

Touching the cleansing of brick walls, I fear there are cases in which the brick walls of compounds do require something to be spent upon their maintenance. I have unhappily not shared Mr. Schmitt's experience, that "good bricks can be had for the same price as bad ones," although I have found in some parts of the country that even bad bricks are not to be had, except at the price of good ones. It may, I hope, be assumed that none of us use any second class material when first class material is procurable.

With regard to the cost stated, I am prepared to give and take a little, but not much. Our compound, as it stands, cost about £3 10s. per boy, but as it includes some reconstructed buildings, I have not the exact figures for a compound of all new construction. I consider, however, that similar accommodation can be supplied, all new construction, for £3 15s. per boy in the case of a compound such as the one in question. The suggestion that increased cost of construction may be regarded with composure, when it results from the preferential use of local material indicates a fine public spirit, but it involves an important question of general policy.

I hardly think that Mr. Schmitt would seriously maintain that an engineer is justified under any circumstances in increasing the cost of any piece of work to benefit a local industry, at the expense of his company, without first securing the shareholder's approval of this generous policy.

The President: Before proceeding to Mr. Tom Johnson's reply to the discussion upon his paper, Mr. Rowland will, with Mr. Johnson's permission, read a contribution from Mr. J. M. Phillips.

NOTES ON RAND MINING.

(Read at March Meeting, 1908.)

By Tom Johnson (Member).

DISCUSSION.

Mr. J. M. Phillips (*Member*): Prof. J. Yates in referring to that part of my contribution wherein I state, "but I prefer underhand stoping for machines, as there you have better footing for rigging up, and all the holes are water holes, which one cannot have either in overhead or breast stoping," says, "that, of course, is not correct." I did not mean that *no* water holes could be drilled in either overhead or breast stoping, but that by drilling underhand all the holes are "water," whilst in "overhead" or "breast" stoping the stopes cannot be kept in proper order unless some of the holes are drilled "dry."

From practical experience I have found that in "underhand" stoping 100% of the holes are "wet," in "breast" stoping 95% are "wet," in "overhead" stoping—six-hole bench—7[illegible] may be wet, but with a four-hole bench only [illegible]% can be reckoned on.

No matter how well a hole may be placed it cannot be absolutely relied on to break according to one's intention. One may "kick" far beyond expectations whilst another may fail to bring its burden, or there may be a missfire, and thus your ideal stope is knocked out of shape; and even in breast stoping it will be found necessary to drill one or more dry holes to bring it back to its proper shape again. Even if it were possible to drill every hole wet, which I contend is impossible in practice, it is not always economical to do so, and although from a health point of view every man should drill wet holes where ever possible, yet if a man is on contract he will drill dry holes, if by so doing he can break more ground, and nothing less than the "sack" will stop him.

I am sorry that the illustrations showing how an overhead stope can be worked by wet holes only is not reproduced in the *Journal*, as it would be very interesting and perhaps instructive to myself and those other outside members who cannot attend our meetings.

I agree with Prof. Yates in saying that it is very unusual to see a stope being worked by drilling all the holes dry, and I cannot think that the cases are as many as Mr. Coombe states.

Mr. Tom Johnson (*Member*): Several of the contributors to the discussion on this paper expressed the opinion that in writing it I intended to wake up some of the members. I shall feel amply repaid for my trouble if I can justly claim to have done so. In the paper I left many openings for attack, hoping that members would seize on them and express some opinions which would take some combating, as I believe one can never get value from a paper unless the discussion causes the writer to either prove his words or withdraw them. As several say, the paper covered a wide field, and it would, no doubt, have been better if some of the subjects had been dealt with alone; but before going on my holiday I only had a short time at my disposal, and thought I might as well provide something for members to think about, as it would be beyond the usual time before I could hope to reply. Several of my critics seem to wish to saddle me with advocating two shafts for a property; I don't do anything of the sort. Some also have a habit of dismissing a subject without investigation, as in the matter of boilers underground, for instance, when they are probably without any experience of it. I welcome any critic who will go for me good and strong, if he will bring some argument forward.

It might interest members to hear of a little job that was done in an upcast shaft at Strangeways Hall Colliery, Wigan.* This upcast shaft, with a furnace situated in a drift at 1,200 ft. from surface, was enlarged from 10 ft. to 15½ ft. diameter whilst being available to wind coal or men. The shaft is 2,163 ft. deep, and the top 1,620 had to be enlarged. Coal winding took place 8 hours and stripping 16 hours out of the 24—four seams being dependent on the shaft for ventilation. Air for the sinkers was carried down in two 18-in. dia. pipes for a distance of 300 ft., below that no pipes were used, the fresh air travelling down the side of the shaft and returning up the centre with the ventilating current from the four seams. I wish to draw attention to the manner in which the ventilation of the portion being sunk was effected. Here we have a heated column of air rushing up the centre of the shaft and a fresh-air current going down the sides of the same shaft. At the time the paper was written the fresh air was travelling down the 300 ft. of ventilating pipes and 368 ft. free. This natural brattice can be found in some of our incline shafts which are being sunk with hammer boys. I mention this work to show what can be done in upcast furnace shafts.

Prof. Yates thinks I am wrong in comparing Kimberley and the Rand, because of our compound shafts and the many loading stations. I am far from agreeing that the compound shaft is the best, as may be seen in my reference to bends; also, I claim that we are wrong in using so many loading stations at one time, for so many loading stations are surely not necessary if, as Mr. Saner mentions, a level is only in use for a year or so. If compound shafts are the best, I am wrong in my comparison, as I do not think the same amount can be pulled from a compound shaft because of the bends and the several loading stations. In dealing with questions like this we must look to, not only what has been done, but what is being done and is proposed to be done in the future: it seems to be forgotten that shaft hoisting, under as good conditions as at Kimberley, is quite possible at several places here on the Rand, and I think it will surprise many of our members to know that there are not many of our hoisting engines that run more than 25% to 30% of the available time, even amongst those that are supposed to be doing a lot of work. If any one doubts this statement, let him time a few engines and see how long it takes to hoist a trip, and then get the average number of trips per 24 hours; he will learn something about the waste of engines. Winding costs are over 1s. per ton, and I am of opinion that some of this can be saved by giving our hoisting engines more work to do.

It gave me great pleasure to read Mr. Edwards' contribution to the discussion, as I like to see people alive to the supposed dangers of any new procedure. The danger of flushing in tailings is not so great in the proposed system as he believes, for what I am proposing is, I believe, quite different from the case he quotes. I am alive to possible trouble, but the manner of carrying out the work would provide the means of dealing with it. I note in the July *Journal* that the late Mr. Gluyas tried the dumping of tailings into the stopes, but this is quite a different thing to water stowage.

---

* Trans. Inst. M.E., vol. xxxv., part 3. Enlarging upcast shaft from 10½ ft. to 15 ft. dia. whilst available for winding men and coal.—Mr. Chas. F. Bouchier.

Mr. Phillips thinks that to hoist an average quantity of 5,000 tons daily through two compartments from a depth of 3,000 ft. is not possible, but he is wrong, as it has been proved that greater quantities have been hoisted from less depths. I am of the opinion that we on the Rand can, if we try, beat all known records in hoisting rock. If Mr. Phillips had started with a useful load of 8 tons, say, instead of a puny 4 tons, he would have found there was plenty of time. If it is thought that a load of 8 tons, plus skip, say, 4 tons, is too great, I would point out that there are plenty of instances in practice where greater loads are on the rope end. The question of hoisting rock wants looking at, not from the point of view of what we see being done here on the Rand, but from that of the best practice elsewhere. As we are able and willing to show others something worth while copying, so we should copy any improvement that others can show us, when it is suited to our conditions; we must not wrap ourselves in any cloak of prejudice to the detriment of the industry. When we know of instances of 3,000 tons of coal being hoisted in 10 hours, are we to sit down and say it cannot be done here when we have better facilities for doing so now opening up. What is possible for others to do, is, given as good conditions, possible for us to do.

Mr. Phillips asks how is it possible to handle the tools, men, etc., in addition to the rock? He says he has been on mines where it takes a separate engine to handle men, tools, etc.; so have I, and for that reason I fancy I know a little of what I am talking about. In many places we still hoist men at midday, we do not do it during the night shift, is it therefore necessary? Is there not also too much time spent in handling material, drills, etc.? I think underground furnaces, machine repair shops, detachable bits, etc., will make it unnecessary to keep engines for this odd work; there is too much odd winding allowed because we do not feel it with so many engines around. A changing gear and a couple of spare skips for handling drills would be a great help to many mines at the present time. If we get away from some of our old ideas and better our system of working we shall not want so many persons for the work as we use at present.

As to handling men, I have known the skips changed and 1,000 persons put down in $1\frac{1}{4}$ hours with one pair of engines here on the Rand.

I figure out the problem of hoisting 5,000 tons of rock, handling the men, examining ropes, etc., with one pair of engines, depth of hoist 3,000 ft. in the following manner:—

Quantity 5,000 tons from a depth of 3,000 ft.; rock 8 tons, skip 4 tons, maximum speed 3,500 ft. per minute, 15 seconds acceleration, 10 seconds retardation, 64 seconds running time and 26 seconds loading time, or a total of 90 seconds per trip, 40 trips per hour, giving 320 tons per hour, say, 16 hours for rock hoisting. Say, number of persons per shift 2,000, 50 persons per trip equals 40 trips, maximum running speed 3,000 ft. per minute, 13 seconds acceleration, 10 seconds retardation, running time 72 seconds, loading 48 seconds, making a total of 120 seconds per trip, giving 30 trips per hour, equals 1 hour 20 minutes each time of raising or lowering the men, equals 5 hours 20 minutes for the two shifts, leaving 2 hours 40 minutes for daily examination of ropes, with Sundays for weekly examination. If more persons are needed, the number per trip could be raised to 80, which in the same time would mean 3,200 lowered or raised. These times can be improved upon as the rock loading ought to be done in 15 seconds, and men loading in 30 seconds per trip. I have not worked out whether the above conditions would give the greatest useful effect for the engines, but I do not think they are far from what would be good practice, but a little heavier load and slower maximum speed might be better. Under the above conditions the extra load due to acceleration, if uniform, would be about 3 tons for the rock hoist.

The ropes for above work may be either parallel or taper, running on a spiral or spiral-cylindrical drum or running on a parallel drum with a balance rope. It must be understood that with a parallel drum and taper ropes the balance rope could not be fastened to the cages, but would be run separately by a light rope running on a Koepe pulley fixed on the drum shaft, the balance rope being run in any part of the shaft that was most convenient. With a spiral or spiral-cylindrical drum there is great weight but no tail rope is needed; with parallel drum and taper ropes we have a lighter drum, but also the weight of Koepe pulley [illegible] nning rope for balance rope, and balance rop[illegible] avy enough to balance its own hoisting rope, [illegible] the rock hoisting rope. With parallel rope and drum the balance rope can be fastened to the bottom of the cages or skips, this makes the changing of cages and skips more difficult. If it is considered preferable to use the parallel rope and drum, probably with a flat rope as balance rope, and providing means for facilitating the changing of cages and skips, then the rope would be locked coil, $5\frac{1}{2}$ in. circumference, $1\frac{3}{4}$ in. diameter, 45 lb. per fathom, weight 11·25 tons, total load 23·25 tons, breaking strain 188 tons, and factor of safety 8; the drum 17 ft. diameter, grooved

$1\frac{7}{8}$ in. pitch, the ropes to chase across the drum. The breaking strain I have taken from a maker's table, the long tons of which I have converted into short tons, giving a reduction of strength of about 10·7% for difference of table strength and assumed actual strength as used in the example. The factor of safety of 8 for new ropes going on is high enough for these heavy ropes.

I do not know if the Government Inspectors would think favourably of a reduction of the factor of safety for these heavy ropes, but I think if pulleys and drums of proper size were used it would be quite safe to reduce it. I fail to see why, if a factor of 6 is sufficient for the lighter ropes in use at present, that 5 should not be sufficient for these heavier ropes when taken off. The following example will show my reasoning. Assume we have two ropes of the kind used for the above example, one $5\frac{1}{2}$ in. circumference, breaking strength 188 tons, factor of safety 8, load 23·25 tons; this rope is used until the factor of safety is reduced to 6, and there is still a strength of 141 tons left in it; and the other a rope of $3\frac{7}{8}$ in. circumference, breaking strength 90 tons, factor of safety 8, load 11·25 tons; this rope is used until the factor of safety is reduced to 6 as before, the strength of the rope being 67·5 tons; why should we require so great a difference in the strength of the two ropes when taken off? If we were allowed to work the larger rope until the factor of safety were 5 there would still be a strength of 117·5 tons left in the rope, surely a safe enough margin; under these conditions would the dangers of hoisting men be greater than now? This principle is carried out in testing the strength of boilers; boilers working at pressures up 75 lb. are tested to double working pressure, but a boiler for 150 lb. is tested to 225 lb., that is 50% over working pressure.

I should prefer to use the parallel drum with taper ropes and Koepe pulley, as, although it is heavier than the parallel drum and parallel ropes, it is the handiest for changing skips and cages, also the sump is clear under the skips, in this case the ropes would be of the following sizes:—

Lower 1,000 ft., $4\frac{7}{8}$ in. circumference, 23·5 lb. per fathom, load $12+2$ tons, breaking strength 112 tons, factor of safety 8; second 1,000 ft., $5\frac{1}{4}$ in. circumference, 27 lb. per fathom, load $12+2+2\cdot26=16\cdot26$ tons, breaking strength 134 tons, factor of safety 8; upper 1,000 ft., $5\frac{1}{2}$ in. circumference, 29 lb. per fathom, load $12+2+2\cdot26+2\cdot42=18\cdot68$ tons, breaking strength 147 tons, factor of safety 8; the drum to be grooved $1\frac{7}{8}$ in. pitch, the ropes to chase across the drum: the balance rope would be 12·0 ton weight, run by a locked coil rope of $3\frac{1}{4}$ in. circumference and a breaking strength of 80 tons; 12 tons gives a little overbalance which is good. All hoisting ropes to be subjected to Messrs. Vaughan and Epton's fatigue test in addition to ordinary test before being used.

I am very sorry if I led Mr. Phillips astray with regard to rail guides, but in speaking of rail guides and rope guides at the same time I thought it would be understood what kind of rail guides were meant, as they are common enough in these times; I meant rail guides as used in collieries, the slippers on the cage encircling the head of the rail; I said two guides per skip, but four would be better. I am afraid Mr. Phillips has not read the paper carefully, as he charges me with inconsistency with regard to the two shaft sinking. I said, if on a property two shafts had to be sunk I would sink them together, meaning such a property as is generally understood, say, 200 to 300 claims; in the other case I was speaking of two shafts going down on the incline, each serving several properties as mentioned: these shafts would have the development drivages going off from them all the time, as is the present practice: the winzes, of course, would be the second outlet, facilitating ventilation. This, to my mind, is quite a different proposition from the other, and I cannot take any blame for inconsistency because he has failed to see the difference; he seems to have missed the point altogether in this matter of incline shafts, and I will take a few points he mentions and try to prove to him how easy it is to be mistaken. For inclines for great depths he fails to see any advantage of the incline over the vertical for the following reasons:—(*a*) more footage for incline shafts; in the case I cited most of the footage is already partly done, or if a complete new shaft was to be sunk under the footwall of present stopes, there are so many points of attack open that the footage costs would be no greater for the greater incline distance than for the necessary vertical distance (also see below); (*b*) more timber and ropes; I do not think so, in most inclines I know of there is a great deal less timber than in the vertical, as to ropes, it is a moot point; (*c*) wear and tear of rope greater; like (*b*): (*d*) extra costs of rolls for rope; (*e*) heavier and closer timber; no; (*f*) loading stations at every two or three stations at least; does Mr. Phillips seriously mean that, if he were to sink a deep shaft, say, 3,000 ft. he would compound it; I believe not. Would he not have to go on the incline in some manner and use loading stations? (See Fig. I.)

As is well known, for these deep shafts the companies start close to the northern boundary to sink, as at (*b*), Fig. I., and reach the reef at (*c*), and may sink for another level and put in permanent loading station, then they have to go on the incline: how many stations are saved by doing this instead of coming down on the incline from (*a*)?

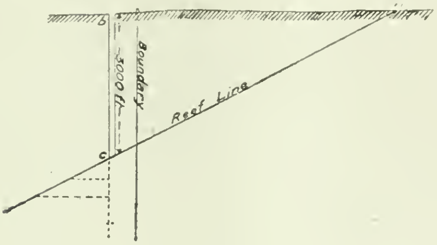


FIG. I

Taken from a purely mining point, the incline method of attacking the ground mentioned actually requires the shortest footage (see Fig. II.).

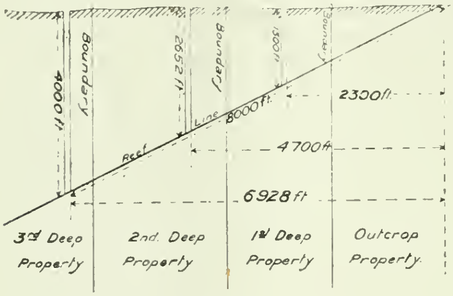


FIG. II

In the sketch we have the surface pegged out by the Outcrop Co., The First Deep Co., Second Deep Co. and Third Deep Co.; the Outcrop Co. sinks its incline and the Deep Cos. their verticals. Now, if instead of the three Deep Cos. sinking vertical shafts the incline shaft had been carried on large enough to serve the several companies, which would take the greatest amount of sinking?

Let us calculate the footage that is done by the four companies and what the footage would be in my scheme of incline work taking the dip at 30°. Say, the Third Deep sinks its shaft just inside its northern boundary and takes 4,000 ft. to reach the reef, the distance from outcrop being 6,928 ft.; the Second Deep sinks its shaft, say, 4,700 ft. from outcrop, and takes 2,652 ft. to reach the reef; the First Deep sinks at 2,300 ft. from the outcrop, taking 1,300 ft. to reach the reef; this gives us for the three deeps 4,000 + 2,652 + 1,300 = 7,952 ft. without outcrop. The distance on the incline to reach the point where the Third Deep shaft gets the reef is 8,000 ft., not much difference, is there? especially when one comes to think of the fact that practically the whole of the 8,000 ft. of incline shaft sinking is done in addition to the vertical work; this latter fact seems to have escaped Mr. Phillips, as it has done many others. The shafts are still open to the danger of subsidences, but are not vertical shafts liable to subsidence if the pillars are taken from about them.

As to whether I am right or not in saying there is too much timber used in our shafts is, of course, a matter of opinion, but there are a few miles of timbering in our shafts where nothing but blocking touches ground; could not the wall and end plates be lighter in these cases, and could not the distance centre to centre be expanded a little. At 6 ft. centres timbering is costing, I should say, £3 per foot in the seven-compartment shafts, and if we put them at 9 ft. centres, which could easily and safely be done for nine-tenths of the distance, something would be saved on timber alone, besides wages and time.

Long Incline Hoisting.—I said, "using endless ropes and cars on inclines a smaller shaft is permissible for a maximum tonnage," this is true on short inclines, and there is no doubt that better work with the present tonnages could be done with the endless rope system in many of our incline shafts than with skips. On long inclines, with big tonnages, skips would probably pay best because of the expense of handling. I expected someone would have asked, if, as I say, 5,000 tons could be hoisted and men handled in two compartments, 6 ft. × 5 ft., of a vertical shaft, what could be hoisted from the same depth through an incline of 30° dip, and what size would the compartments be? The same amount of rock could be hoisted, the unit amount handled being greater. The load would be: rock 12 tons, skip 7, rope 16, and friction 7, equals ·42 × ·5, equals dead load 21 tons; rope 5 in. circumference, breaking strain 155 tons, factor of safety 7·4, maximum rope speed 4,000 ft. per minute, acceleration 10 seconds, retardation 10 seconds, time of running 100 seconds, filling 33 seconds, making 133 seconds per trip, 27 trips per hour or 320 tons, as before, the size of compartments being 6 ft. wide by 7·5 ft. high from sills. These large total loads are nothing uncommon as I know of instances where a load of 18 long tons* was hung under the capping.

Mr. Saner asks how it is that in other places they are sinking large shafts for smaller quantities and shorter distances than I mention in my paper? If he had read my paper carefully he would have found that I said "the amount of rock to be hoisted should not be the measure of the size of the shaft, but that the amount of air needed for the number of persons to handle the rock should be the guide." I particularly drew attention to the fact that the three compartments mentioned were not large enough to pass the necessary air without an extravagant waste of power, clearly implying that I did not consider such a shaft large enough; as a matter of fact, I never calculated what the necessary size of shaft should be, as that has no place in this paper. Mr. Saner quotes the Bentley Collieries' shafts as being 20 ft. in diameter for 2,000 tons per shift, but he forgets to mention the fact that one pair of engines is to be used for this work; if we were only doing as well it would be a great deal better than at present. How many cases from the whole of the mines of the Rand can he quote me of where two pairs of engines working two shifts per day are doing that amount—not many, I am sorry to say. I do not know why the Bentley people put down their shafts 20 ft. in diameter. There are several points that they would have to consider, two vital ones being the size of cage to carry the cars holding the coal, and the amount of air necessary to provide an adequate ventilation; now we know that for either of these reasons a colliery shaft would need to be larger than a gold mine shaft for the same tonnage, for the bulk of coal is greater than rock and has to be carried in cars, also the amount of air necessary is greater to carry off the gases liable to be met with. If giving an opinion why those shafts are 20 ft. in diameter I should say it is for one of the above reasons, but even if their shafts were too large for the work, that is no excuse for our doing likewise.

---

* Winding Ropes and Capels. W. Routledge, Trans. I.M.E., vol. xxxiv., part i.

Mr. Saner thinks the comparison of Kimberley and the Rand absurd; not hardly so bad as that; he, himself, is manager of a shaft where the hauling will, or should be, under as good conditions as Kimberley, and how much rock is it proposed to hoist through the six hoisting compartments with the three pairs of engines? Also, take the South City, for example—if 1,000 stamps are erected the required tonnage of rock will be about 12,000 tons per working day; then three sets of engines should do that, and handle the men, tools, etc., that are necessary for the work. At these and other shafts there is only one station to hoist from, and the width of seam or reef, dip and facilities for handling the rock under ground do not enter into the question: if two engines are put down to do the work that one can do, it is no use talking of width of seam, dip, etc. If the rock cannot be got to the box it only makes matters worse; there is some excuse for having a little extra power to handle the rock one can get to the box, but to put in engines to hoist rock one cannot get to the boxes is madness.

Mr. Saner says mining banket is not mining coal. I have done both and agree with him; but in hoisting who has the best of it—the man who handles the greater volume for the same tonnage and a product that suffers by being knocked about, or the man who hoists the lesser volume, say, four-sevenths of the other, and a product which benefits by being knocked about? Consider the shaft area occupied by the cages to hoist 4 tons per trip in both these cases; in the Bentley district the cars would hold about 1,000 lb. each, and each would be, say, 3 ft. × 2 ft., and eight would be required, four per deck, 6 ft. per car, say, 24 sq. ft. area without clearance; for rock we should take a skip 2 ft. 6 in. × 4 ft., say, 10 sq. ft., say, about two-fifths the space (see Fig. III.). It is something of this kind that troubled the Bentley people. Mr. Saner thinks it folly to spend money in elaborate arrangements when a station is only to last a year or so; well, I have seen 26,000 tons come off two levels in a month, and some of our members think three levels are not enough for a 200-stamp mill, but if, as he says, a level will be worked out in a year or so, I think there is much better work being done than some of us are aware of. As to the cost of fitting up boxes, I think money spent on boxes, within reason, is a good investment. I know I have been thankful for box room often enough, and the way I look at it is how much per ton will it cost and how much am I likely to gain. Assume that on a mine with 250 ft. backs it is decided to put in boxes only at every second level; at the odd levels a crosscut has to be driven to such a point that a pass going down at 55° will strike the box of the level below; how much extra money will it cost in comparison with making two boxes, and when you spread it over the tonnage of two levels of a mine 5,000 ft. long with 12 ft. of stoping width it comes out at $\frac{1}{100}$d. per ton for every £100 spent extra.

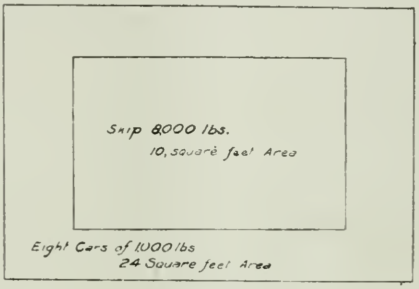


FIG. III.

Mr. Saner thinks I am wrong in saying that we are not in the front rank in mining matters, because we have some of the finest mining men in the world here. It does not matter, to my mind, how good these mining men may be; they are human like the rest of us, and liable to have the same failings. I do not think they claim to be infallible, nor do I think they should be, they would have no business here if they were. I fancy there is room for improvement, and that improvements can be made, especially in our methods of handling rock, so in my own clumsy way I am trying to point out where I think something can be done. Mr. Saner says, given plenty of room above the collar, accidents should be few or none with good drivers: but what about room above the tip—it is there we require a little room, and most of us are loth to leave room there, as we want all we can get under the tip. Considering the number of trips an engine has to make during a shift, the weight to be set in motion, the velocity to be attained, and the short time between starting and stopping the load, I think it is expecting too much from the engineman to go on year in and year out without an accident, no matter how good a man he may be.

I have long been in favour of applying to coloured labourers the no-work no-food motto, but have always seen the difficulty of cheap food, as I do not think the Government would allow the companies to control the sale of food to the coloured workers, as it savours too much of truck.

Mr. Weston says Prof. Yates has pointed out that it is a different thing to hoist rock from one point than from eight or ten; it may seem strange to Mr. Weston, but I am quite aware of the fact. The point that requires elucidation is whether the large number of points are necessary: I believe not, and suggest for the main hoisting, one: for the subsidary or secondary, as few as possible—say, three on most mines. I know of many places where there are two shafts—one shaft having a rock hoist, and the other a rock hoist and a man hoist—neither working 30% of their time when they ought to be running 60% to 70%; this is not done because of faults, dykes, etc.

As to reading technical literature, I am pleased to say I do, and have done so for many years, and if Mr. Weston had done so he might have had a nodding acquaintance with some of the points raised in my paper. His remarks *re* circular shafts are not practical, and I would strongly advise him to pay a visit to one in process of sinking; and I hope there are not many more in the position of wanting to know, in these days, how rope guides are fastened in colliery circular shafts: there is one thing, they are not left hanging like the end of the bell line. As to the colliery arrangements reposing on the bottom of the shaft, I fancy the colliery people manage to get through without this happening.

As to handling rock on inclines, he has something to learn, but perhaps he has not had the opportunity of seeing cars hauled up a 45° incline as I have. He seems to have a vivid imagination of the mess there would be: I should expect an imagination like that would help him to devise means for preventing the mess. I know steam boilers underground are not so greatly in favour in collieries as they used to be, there being two very grave dangers in coal mines to guard against, that is danger of firing the coal and explosions of firedamp, dangers that are not present in our gold mines. There are some still in use, and as far as I can gather could not be beneficially changed on the score of economy. At one colliery I know of there are underground three Lancashire boilers, 28 ft. × 7 ft., which were taken down whole, except for the mountings, and have proved very satisfactory. Mr. Weston speaks of properties being divided by a fault or dyke with 500 to 1,000 ft. throw: but this is quite the exception, and when things like that occur the owners have a very sensible way of re-arranging claims to nullify the effects of such faults. He asks, have I never heard of bratticing a shaft? I have very often, and I know enough about ventilation to know that brattices are only a makeshift, and are of little use after starting away from the shaft. Has Mr. Weston never heard how hot our deep shafts are before they are connected, and how glad everyone is when the connection is effected. There are a few people going to learn something of ventilation when we get a handy electric drill going in our mines. Several ask me, could I develop as much ground from two shafts close together as I could when they are the usual distance apart. No, I could not, but if Figs. IV. and V. are studied a little, it will be found that I should not make a bad show, compared with the system of opening out that we are used to. Mr. Weston wonders if I am poking fun in suggesting ten machines on a face, I am not; I believe the time will come when it will be necessary to use large numbers of machines on each face, as the faces will have to advance much more quickly than now or they will not be kept open.

A short time ago Mr. Richardson read a paper in which some very useful points were discussed about the size of pillars and the weight they would carry in deep level mining; I am not sure whether I am quite correct or not, but I believe the general feeling was that we should not be able to leave pillars enough and work profitably at the same time; my idea is that we shall have to clear out everything and pack, if possible, moving the faces along at a fast rate. If the system of using more machines for each face is slowly introduced, I believe it will be found to be no more expensive than our present way of doing it; anyhow, it would not cost much to pick out a good worker who could be trusted to try the scheme fairly, and let him start in a stope not far away from the shaft, with three or four machines and helpers, and build up to the full amount of machines that could be worked on the face, to find the limit. I believe it will be found that four benches per machine, as sketched, will be enough, for I do not think a machineman can be found, running two or three machines, who keeps four benches per machine. If more machines are used on a face it is reasonable to suppose that a less number of faces would be necessary for the same tonnage, also that any particular face would move faster and the block of ground be exhausted sooner, so that the blocks may be left larger, which would be done by either having the drives or winzes, or both, farther apart. Then as to brake roads, shaking shutes, etc., being more efficient, if more rock is broken more will need to be removed in the same time, and moving the greater quantity in the same time, there should be a reduction in the standing charges. It costs the same to fix up the rails at the face end of a brake road, or a shaking chute, for a small amount as for a larger amount, also if plenty of rock is about the labour force accomplish more and the gear is more protected. It is all a question of keeping down the unit cost.

Mr. Weston also asks what would the miner in charge of the face do if he found one workman drilling more ground than another? He must either sack the miner or the stope would get out of shape. Perhaps he would allow one of the above things to happen, but, fortunately, a miner would have the common sense to arrange that the best drillers would be put where the most drilling was necessary, as is done now by both machinemen and hammermen, if they know what they are paid for. As to handling machines, etc., I have always noticed that the good men are mostly those who move their gear the least distance, generally a bench or so away. He also thinks a machine ought to have 80 ft. of face to work on, I cannot see why; if a stope is in benches I fail to see why if a machine, call it No. 1, is on a bench, the other benches should wait for No. 1 machine; what is the reason the other machines should not be used on some of them if they can be broken with advantage? if the same mind is directing the whole of the machines the benches may as well be broken in a shorter time by the use of more machines instead of waiting for No. 1 to be free. Speaking of the diagrams of stoping holes, he says he would be under the necessity of sacking a man if he found him with holes placed as I show them, and I fully believe he would, judging by his remarks on mining. He is not the first to be deceived by the look of the second hole to go having apparently little to do, it is that little to do that counts. I have often seen men try his system and be quite surprised to find the holes looking at them when they got back to the face. Mr. Weston must have been more fortunate than I, and some others I know, in the ground he has worked in, for I have seen men in low stopes drilling on his system on contract and not making wages, and when they have been placed on day's pays and made to drill as sketched, in less than a week they wished to go on contract again. As to it not being necessary to fan back holes a little, how is the bench to be held to size in most cases? the machine is wider than the hole and the bench breaks off at the hole, so if the hole does not look in a little the bench is smaller at the back than at the front; if Mr. Weston has been used to ground that breaks back I must again compliment him on his good fortune, or is that the reason why a machine wants 80 ft. of room so as to form new benches as the others get too small? In some cases where the bench is wider than the amount to be drilled then there is no need to fan the back holes. If the amount of ground can be broken in a 38-in. stope, as he implies, which means practically a bench 6 ft. across for five holes, I am wondering why breaking costs are not lower. Mr. Weston thinks handling the machine under the arm is wasting time; he is not alone in that, but that does not make him right: where there is no parting in the foot, good stoping cannot be done without putting the machine under the arm for the bottom holes, and the man who does not put the machine under does not know how to handle it, or is troubled with the fault of many of our Rand miners—tiredness. The chisel bit has the four reaming edges he speaks of. Several critics think I am wrong in having used 2½d. and 3½d. steel. I used 2½d. steel for the 8 ft. drills and over, and for shanking; and 3½d. steel for the other drills. Some time ago a paper was read before the Society on safety fuse, cheap fuse being condemned in the same manner that cheap steel is being now. I found the price of fuse go up, but should not like to swear that the quality did.

Development from two shafts close together, or from a single shaft (see Figs. IV. and V.).—If in sinking, No. 1 shaft gets to the reef first, start the second and first crosscuts, then as soon as the second shaft gets to the reef, start driving both ways until points of first winzes are reached, say, 400 to 500 ft. distant. Start off these first winzes as development shafts, say, 10 ft. wide, opening out the levels from these; if at any time it were found that these did not give length enough between, then the next winze on one side, or both sides, could be used, but I do not think it would be necessary; in the meantime carry on the sinking of No. 2 shaft until bottom is reached, then start number three and four crosscuts, cut stations and ore passes; the main incline would be started from number three and carried on, a temporary box being put in there. As soon as number four station was ready, and connection made with number four drives, the development rock from fourth and below would go to number four station, and when ore passes were ready the rock

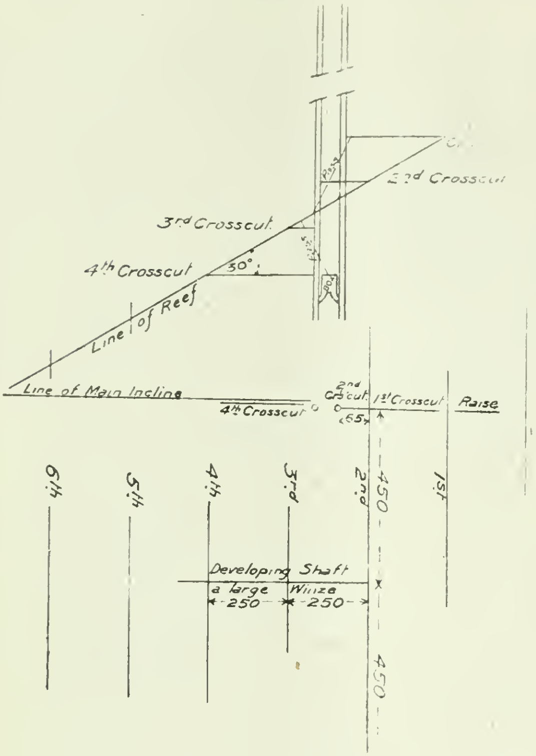


FIG. IV.—SECTIONAL ELEVATION.

FIG. V.—PLAN.

from upper levels would be handled there, and number one sinking, etc., could be finished. If No. 2 shaft gets to the reef first, go below a little and then start Nos. 3, 2, and 1 crosscut; when reef was reached on No. 3, start driving both ways to point of first winzes, and start the development shafts and main incline; carry on sinking at No. 1 shaft to bottom, put in box, cut station, and drive No. 4 crosscut; when connection is made with the fourth drives, and the boxes, etc., finished, turn development rock to this shaft and finish sinking No. 2. It will be seen that cutting stations does not interfere with the development, and it is the same in the main incline, the development will go on without interfering with it, nor will it suffer whilst stations are being cut; the development rock will go up to a station above through the development shafts or winzes; as portions of the shafts or winzes are freed by the hoists being advanced, such portions become available as faces to commence stoping from. In developing from a single shaft, as soon as room for number three crosscut is attained, start it, and when reef is reached drive on it as before and start development shafts or winzes, using one pair of skips to handle the development work, and the other pair for finishing the sinking and cutting stations and boxes and crosscut at number four. If, as it should be, the higher side company had driven their incline down, the higher levels could be started also, the rock coming down to number three box.

Mr. Lane Carter thinks a good electric bell system is sufficient in the majority of cases, but I am still of opinion that telephones are a paying proposition, and I also strongly advocate a speaking tube from banksman to engineman. With regard to rectangular versus round shafts, he brings forward the old stock argument of utilisation of space; this utilisation of space has cost the Rand some thousands of pounds, and is a poor argument, as I hope to prove. In my opinion, the reasons why we keep to the rectangular style of shaft are:—(*a*) Use, it being second nature to use them; it would naturally be the form chosen for sinking on the reef, so as to have as much reef in the shaft as possible, not only to ascertain the value, but also to help to pay expenses. (*b*) The facility for lining with timber, which is generally the most handy thing to be got for the purpose in new countries. (*c*) What I call the mistaken idea that we want so much room for hoisting purposes. (*d*) Being able to turn bends from them and follow the dip of the reef. For the last few years I do not think that any of these reasons should have been allowed to carry any weight as argument against round shafts, as:—(*a*) In all our vertical shafts we have had to go through country. (*b*) Where lining was necessary we have had the choice of other material than wood. (*c*) All the space of a shaft should not be taken up by the hoisting arrangements, it not being necessary and certainly not good policy. (*d*) In some cases bends have not been put in, and in many others where they have, it would have been much better if they had not. I believe, myself, that the question is simply one of cost, and that a round shaft would come out cheaper than a rectangular one. It should be sunk faster, as it has a much larger sumping area and has less periphery to be dressed down. I am glad to see he agrees that there is a chance for endless ro[illegible]ork underground on the inclines, and I am s[illegible] that if the system is introduced it will be found good policy to adopt it both on the rise side and dip side of vertical shafts. I agree with him that electricity will be very good for underground work, if we are to make extended use of it above ground, for it is not the best policy to have several different methods of power supply on the same place. With regard to power schemes, I have often wondered why some of our large firms have not taken a colliery, turned the coal into gas, generated electricity at the colliery, and transmitted the power along the Rand.

He does not think there are many mines along the reef that can supply a 200-stamp mill off three levels; there are quite a few if we can only get rid of some of our old ideas. A 200-stamp mill, milling 7 tons per stamp per 24 hours would, with 20% sorting, need 1,000 tons per shift broken in the mine, and 1,000 ÷ 3 equals 333 tons per level. Say, the property is 5,000 ft. long, of which 4,000 ft. is open, with winzes every 500 ft., giving 16 faces per level; 333 tons ÷ 16 gives, say, 21 tons per face, which on a 4½-ft. stoping width means a little over 1⅓ fathoms per face; this is for one reef with a 4½-ft. stoping width, and I think the greater part of the mines with 200 stamps have more than 4½-ft. stoping width.

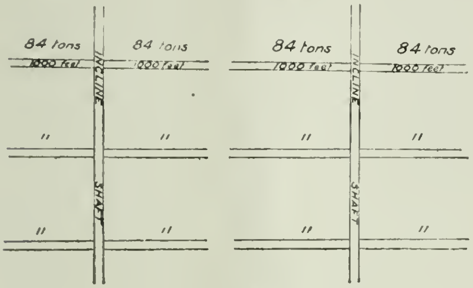


Fig. VI.

Now we have two shafts with the three levels—(see Fig. VI.)—open on each side, making twelve short levels; 1,000 tons have to come off these twelve levels, which equals 84 tons per level, and with, say, nine hours' tramming, there is 9⅓ tons per hour from each, not too much to expect, I think. I cannot see why this could not be done in most mines, or why in the hoisting problem we should pass remarks about having to haul from nine or ten stations. It is known to all of you that things are being done on the Rand now that many people two years ago would have been ready to swear could not be done, and we must always remember that the impossibilities of a few years ago are the common occurrences of to-day. I agree that the contractor should have a direct interest in saving stores, etc.; the blasting and rigging gang would, of course, consist of one of the contractors, "of whom there should be three, if two shifts are working," with help to clear away the gear whilst he himself would be loading up the holes. With regard to packing with sands, many areas could be flushed full without any gas that might be produced having a chance to injure anyone, notably in the outcrop mines, and these are going to prove dangerous to the lower side mines if something of the kind is not done. There are many areas in a mine where acid water would never come into contact with the tailings, and it must not be forgotten that by packing our stopes we bring about better ventilation to deal with any gases given off.

The system of drilling, as shown in Fig. V. of the paper, I have found to answer very well in avoiding breaking too much waste; it will be found that no hole is fast, and it will not cut into roof or floor; it is principally for places where there is no parting at roof or floor, and where there is a false hanging at a short distance above the hanging of the stope. As regards two classes of certificates for miners I do not thing we shall get much benefit from them. for with one certificate we always have the date of issue for a guide. It is a very hard thing to judge miners; I have seen places where old hands, with years of Rand experience, could not make anything like wages, and where men who had not had two months experience made twice as much money. I have also seen clinking good men in certain stopes become failures in other stopes of the same mine with different dip or width of stope. The system I am advocating of putting more machines on a face would be a complete failure, from want of adaptability, if the different machine runners were allowed to go their own way; but by putting a man in charge of the face who has been proved to be a good man under the conditions ruling there, we ought to get better work done. I think it much better to take the man whom we know to be a good man and use his ability to direct ten or twelve machines, instead of leaving him on two or three, and leaving the other machines in the hands of incompetent men.

Mr. Phillips says truly that the output of a stope is limited by the amount of rock that can be taken away from it, but I hold the opinion that more rock than four or six machines ordinarily produce can be taken out of a stope. Some adopt the principle of splitting up the face into sections for shovelling purposes, but work the sections alternately according to where the rock is; I believe this principle can be carried further and the several sections worked together. He says he has no faith in a rigging gang, but likes every man to rig up his own machine; I find that every man is not competent to do so, and in the way I propose we have a much better chance of getting a good man who will see to the machines being rigged up properly. As to being able to keep plenty of rock in stopes I cannot see why big mills should make any difference; the mills must be supplied; so to get extra rock the expense of a few extra machines should be borne for awhile until a surplus is produced; of course, if a manager has not got a labour force large enough he cannot do it: available rock should be treated like development work and kept well ahead of requirements, as it comes in handy in case of breakdowns. I did not know that I advanced any claim as to sorting and packing underground being new, nor did I claim anything new for flushing tailings into stopes, although whoever starts doing it will be the first here; I believe I mentioned that packing workings by this method started in America in 1887.

With regard to long and short holes, if Mr. Phillips had noticed the diagrams he would have seen that the length of hole varied for different stopes; in the tight stopes the length is 6 ft. He says the average length of hole here is 6 ft. and average burden 2·5 ft. I wish I could believe him, for the average footage drilled per machine was not above 21 ft. and fathoms broken not ·5 on the mines of one of our largest firms here for the year 1907. I am glad there is something we can agree about, for he says that where holes bull-ring he prefers to put a little more burden on and give a little more explosive; so do I. I think it a better paying way than shortening the holes Mr. Phillips says he has seen old miners since the eighties placing holes in the manner I describe, and that the old miners know all there is to know about placing holes; I am pleased to hear it, particularly as Mr. Weston says only incompetent men so place them. I do not think anyone can claim that all our old miners are good men, even if we restrict the term to the old Rand miners; I might have expressed myself better by saying old mining men in distinction to men of little previous mining experience. I did not use the words "the great majority," but, "a great proportion." As to the cut, there is plenty of time to prove it: I do not think Mr. Phillips has seen the cut as described used for pulling to a slip, if so, it is not the best way of doing it.

I am quite in agreement with him as regards an adequate supply of drills, spares, machines, etc, but there must also be adequate control over same. The system of drill supply I have been used to has been to send drills down twice per shift, once in the middle of the shift and then about the end of drilling time, so as to be ready for the oncoming shift. I have tried working men under bosses who have gone through the mill and had to prove themselves able to earn their own living at the work; I do not care where a man comes from, whether it is from the ranks or from a school, so long as he is capable. In dealing with new hands I do not think we should make hard and fast rules, but treat the men according to the diligence and ability shown. Some men would never make miners, whilst others seem to have a gift for mining.

I think Mr. Phillips made a slip in his example of a bench 10 ft. long by 6 ft. wide. If I drilled the four holes to break the bench, why should I only have 2 ft. 6 in. on? I should have slightly over 3 ft. on the front holes and slightly less on the back holes, and if the front holes bull-ringed I should not lose the bench, as I do not practice firing the four holes at one time. In overhand stoping it is not necessary to use dry holes; let the benches dip or run towards the bottom of the stope and then the holes will dip to carry water; this dip should be more pronounced than shown in Fig. III. of the paper. If a hole breaks badly and cannot be rectified by firing again, get a hammer hole in and clear that way instead of drilling dry to gain the bench again. If Mr. Phillips drilled dry holes on a bench with wet holes in front he must fan the back holes, which Mr. Weston says is not necessary.

I start by being in agreement with Mr. Schmitt with regard to different generations looking at things in a different light. There would be no need for papers and discussions if we all looked at things in the same way. When Mr. Schmitt says no one would sink a shaft larger than was necessary for hoisting the rock, he is wrong, for given that all the compartments provided are wanted, there is quite an amount of room provided in some of the new shafts as clearance, so as not to interfere with ventilation too much, thus recognising that ventilating room is required. I think it is cheaper to sink the shaft a little larger than to provide extra ventilating power; taking the three-compartment shaft as being large enough to supply hoisting room for the tonnage quoted, I should certainly sink a larger shaft, so as to provide for ventilation. From the number of persons, Mr. Schmitt requires to handle this tonnage, I should say he is a great advocate of hammer work: but to enable us to start up some fresh work when money comes our way again, we have got to get down to about half his estimate of coloured. There are objections to mechanical truck haulage on inclines of 45°, but I may say I have used it on inclines of 44°.

Mr. Richardson thinks it wants a stronger argument than the absence of record hoisting performances to show that we have a long way to travel before we are in [illegible] front rank. You know the old saying to the effect that some are born great, some acquire greatness, and others have greatness thrust upon them; I think there is a lot of the last about us and that we have got to fancy we have earned it. I do not think we have more than one mine that can show an average of over 400 tons per shift for the hoisting engines; and the large reduction in costs this last 18 months shows that we have had nothing to crow about.

Mr. Richardson gives us some very interesting figures on winding speeds, he also tells us that at Kimberley they have done 363 tons per hour on several occasions, better than many of our shafts do in a shift. Then for British collieries he tells us the speed at Denaby and Cadeby is 2,289 ft. in 55 seconds; he might have gone further and told us that they hoist 269 short tons per hour with a useful load of only 1·48 tons, the total load being 22·4 tons. Bolsover* hoists 221 tons per hour with a useful load of 2·35 tons, Mamfield 336 tons per hour, Rotherham 323 tons, the useful load being 5·6 tons and the total load 20·6 tons, Baggeridge and Sandwell Park 430 tons per hour, useful load 7·16 tons, total load 19·5 tons, H. C. Frick Coke Co., U.S.A., 300 tons per hour, Hitchester Main 300 tons per hour, useful load 4·92, total load 21·67 tons (these are short tons): then they have a bulk 7/4 of ours for the same tonnage and have to carry it in cars. Looking at the total load relation to useful load at these places, and what it would be for us at the same depths, I think you will admit that we have the best chance. I mentioned the total loads at these places because some may think I have fixed the loads in my examples at too great a figure, but if these collieries had to hoist from 3,000 ft., as in the examples I give in answer to Mr. Phillips, they would have to use a much greater total load than they do at present and greater than the load I use. As shown in my example, it is possible to use the balance rope in such a manner that independent trips could be made. Oval shafts would be preferable to round where we wished to use two sets of skips in one shaft. I do not see why we should not use tipping skips with rope guides, by using wood steadying guides at the dump.

Now let us look into the matter of bends and compound shafts and see if we could not do as well, or better, without them. First, I must thank Mr. Richardson for his courtesy in using the Geldenhuis Deep as an example, as he knew I ought to be familiar with that mine.

In Figs. VII. and VIII. I show the No. 2 shaft as it is, also how I think it ought to be. In Fig. VII. there are shown crosscuts driven at Nos. 3, 4 and 5 levels, No. 3 being the top level at this shaft. The bend is turned off immediately above No. 6 level, the 7, 8, 9, 10, and so on, being on the incline. The same shaft could have been sunk as shown in dotted lines. Now, suppose the shaft had been sunk, say, 300 to 400 ft. further south to give more ground on top side of shaft, the distance to the reef would be greater by 200 ft. or so, giving five levels on the top side instead of three, then by sinking another 200 ft. or so we should have the 9th and 10th levels by crosscutting, by doing this it would be as in Fig. VIII. Here a crosscut is driven to what would be the 5th level and a rise sent up to open up the 4th and 3rd, the rock from these levels would be sent to the 5th by self-acting endless rope incline, crosscuts would be driven on the 6th and 7th levels on the north side, and on the south side we should have the 8th, 9th and 10th by crosscuts; boxes would be put in at the 5th, 8th and 10th; that is the way I should get out of having shallow vertical shafts, bends, and avoid duplicating machinery, and also find work for the one pair of engines that would be necessary. When it was necessary to develop the lower part of the mine this could be done with a small pair of engines. In cases like this, development would probably have been done much quicker by taking longer to sink the shaft, as more points are opened for attack. It may be all right for people to talk about saving money by using bends, but I have seen a good few mines using bends, and I fail to see any benefits arising from them, especially with the reef dipping 25° and over, the more the reef dips the less need for a bend. If the Geldenhuis Deep had been sunk, as in Fig. VIII., it would be an easier and cheaper mine to handle, despite the crosscuts shown.

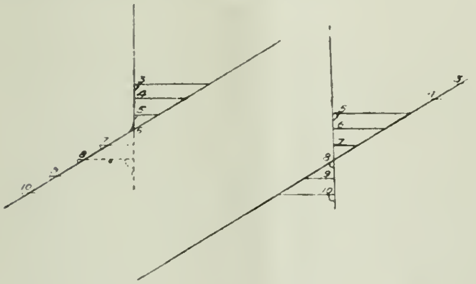


FIG. VII. FIG. VIII.

Bends may be convenient when nothing has to be done beyond playing at hoisting with them, but try to push things and begins to appear trouble without looking far for it. Taking the illustration of the Geldenhuis Deep and Jumpers Deep, and assuming that in both cases the plant is worked to full capacity, which would be the cheapest way of doing the work—by two engines in the present manner or by having laid out the work as I have shown? As well argue that in a 6,000-ft. vertical shaft it is as cheap per ton to hoist by one lift as by stage winding when working at full capacity. Of course, someone will say it is quite easy to talk of it now, but I must say I have held the same ideas ever since I first knew the Rand. The accidents that Mr. Richardson quotes the number of, were not exactly the accidents to shafts, cages, engines, etc., that I referred to; many of them were of men falling from cages, because it is not the general custom at Home to use gates on the cages as we do here.

---

* Twenty years ago, a pair of Daglish's engines made a record run of 246 short tons per hour from 1,125 ft., at Bolsover.

The sinking of two shafts close together would not make ventilation much worse than now, for what is done at present to stop short circuiting between shafts? What difference is there between ventilating 2,000 ft. outwards from our shafts as at present, and ventilating 2,000 ft. outwards from the two shafts close together? In the proposed long inclines I gave alternate methods, but if stopes were packed, there would be no broken roof, even with what may be called a bad roof, as it would come down bodily; the most broken part of a caving is around the edges of the area, and I have seen miles of road open in caved areas where it could not be told by looking at the roof that there had been any caving. I believe Mr. Richardson and I will have to agree to disagree about subsidence. I have seen any amount of subsidence of surface over packed workings, never mind unpacked workings. When in Belgium, I went to visit a pit using the flushing method for packing; before I saw the pit I knew I was close to one, for every house in the village was cracked, and on making enquiries I found the company kept a number of bricklayers doing nothing but going through the village repairing houses. We shall know more about it by-and-bye, for the worked-out area keeps growing, and so does the caving in some of the mines. It is just as easy to keep rock in stopes when feeding large mills as small ones, provided you have the labour force. I hope Mr. Richardson never thought I seriously proposed to stope without supporting the roof; where was I going to put the tailings I used up so much ink in writing about? I made a slight mistake in the diagram of the new cut, as all the holes are water holes parallel with each other; both top and bottom holes are charged and lit together, the fuses being cut for the top hole to go first.

With regard to holidays, I have seen men who did not like taking a short holiday yearly, preferring to allow their leave to accumulate till they could take the long one. In the case of a man going to visit other fields I should, of course, expect the company to offer the encouragement financially, as it would be to their benefit to do so, at least I think so.

With reference to subsidences I hear that Johannesburg has again been visited by supposed earthquake shocks. Now it has been said that it is due to caving taking place in some of our mines, so the mines in the affected area are examined and no caving can be discovered; then some say if that is not so, then it must be earthquake shocks that are felt. It is quite possible, and probable, that there has been no earthquake shock, and yet it is our mines that have been the cause of the shock. With strong roofs, like the roof of our mines, large areas can be opened out without any signs of taking weight for some years, but eventually the roof weight becomes so great that the lower portion begins to rip away from the higher portion and starts "talking," it is this "talking" that is heard and felt; the movement might be so small as to escape notice beyond observing that the loose pieces that hang about some pillars are a little looser. It must be remembered that although the distance moved is so small, yet the weight is so tremendous, it may be a million tons, that we get a shock. I have known of railway trains going over rock outcrops causing vibration in houses half a mile away, and in the case of the above shocks the probable mine should be found by tracing the line of vibration.

This reply is, no doubt, rather loose, but if any member would like to discuss any of the points that may still be undecided, I am quite willing to do anything I can to help him.

The President: We must congratulate Mr. Johnson very much on his reply. It has been a most exhaustive one. He has replied very thoroughly to all his critics, and has raised more controversial points, and he kindly leaves an opening for his critics to make further remarks. That is very good of him, I am sure. His reply has become as valuable as his original paper. He has amplified many matters on which there may have been a good deal of doubt existing, and I think the hearty thanks of our Society are due to him for the excellent paper which he has placed before us.

THE ADAIR-USHER PROCESS.

(*Read at May Meeting, 1908.*)

By Alfred Adair (Member.)

DISCUSSION.

Mr. A. H. Hartley (*Member*): For current slimes on the Rand I think it is pretty well proven, by practical everyday results, that the use of umber does not increase the extraction of gold. I think it will be conceded that all the gold in current slimes that can be economically dissolved is dissolved. The successful treatment of slimes on these fields is therefore a question of washing, or separating the gold bearing liquor from the slimes, the recovery of gold from the solution, and of costs. For rich slimes, from, say, 3 dwt. and upwards, there is a great deal to be said for the Adair Usher process and for filter pressing.

For persons immediately interested in current local slimes I think filter presses at present may be considered much too expensive, both as to original cost, costs for repairs, and ordinary working costs. They are often the direct and irremediable cause of high residues, if sands, however small in quantity, should escape to the slimes plant, or when frames and cloths become worn or broken.

Comparing the Adair-Usher with the decantation process, one can assume that all the gold is dissolved in both cases. (As a matter of fact, days of agitation in contact with cyanide solution with or without aeration will not dissolve the last 2 or 3 grains of gold held in the residues of the average Rand slimes.) Treating, say, a 2 dwt. slimes by the decantation process with two washes of 3½ tons of solution to 1 of slimes, and having a 1 to 1 settlement after each of the two decantations, the residue would contain 0·163 dwt. dissolved gold, equal to 8·15d. per ton. Supposing by using 2 tons of wash to 1 ton of gold bearing solution the Adair-Usher process would completely displace the gold solution from the slimes, and taking the cost for precipitation of gold in zinc boxes at 1½d. per ton of solution, this would come to - 7 × 1½ or 10·5d. per ton.
As against - 3½ × 1½ or 5·25 ,,

Difference - 5·25 per ton.
Add royalty - 2·00 ,,

7·25d. per ton.

To this might be added, possibly, extra cost for pumping. There would certainly be the cost of pumping about 3½ tons of liquid to every ton of slimes treated, in addition to decantation costs. In any case, the extra costs as already noted leave a very small margin in favour of the extra recovery.

These observations, I admit, are somewhat superficial, but as most papers read at our meetings are written with the professed intention of causing discussion with the object of ultimate enlightenment, my remarks on Mr. Adair's paper may lead to the desired result. In the end it is the most effective and most economical method of gold recovery, irrespective of criticism, which appeals to the mining industry in general.

Mr. C. E. Rusden *(Member)*: This is scarcely what may be termed a discussion on the above process, but more in compliance with our late President's request to place before the members some data obtained from the working of the process on a large scale. Therefore I have pleasure in giving you the following from the Geldenhuis Estate & G. M. Co. slimes works.

These figures are for the purpose of comparing the extractions, working costs, etc., taken over the twelve months decantation process directly preceding the installation of the "Adair-Usher" process, and the twelve months following.

During the first period under review 48,064 tons of slimes were treated, containing 4,953·38 oz. fine gold, equal to 2·061 dwt. per ton; out of this amount 4,163·16 oz. fine gold were recovered, equal to 1·732 dwt. per ton, or 84·04% actual extraction. During the second twelve months 59,800 tons of slimes were treated, containing 5,698·05 oz. fine gold, equal to 1·906 dwt. per ton, of which 5,322·295 oz. fine gold were recovered, equal to 1·78 dwt. per ton, or 93·40% actual extraction. This shows a difference of 9·36% distinctly in favour of the Adair-Usher process, and I am sure you will all agree with me that it is a very considerable item when one takes it over the huge tonnage of slimes daily handled on these fields; but what this increased extraction meant to the Geldenhuis Estate was a reccovery of 533·38 oz. fine gold (in value about £2,246) more than we could possibly have expected from the decantation process.

Working costs are also in favour of the Adair-Usher process, as a decrease of 2d. per ton has been effected since its installation as compared with the preceding twelve months. This saving would have been still greater had the costs of installing the process been charged to capital expenditure instead of being included in the working costs. So by these two items alone the sum of £2,744 has been added to the profits of the Geldenhuis Estate during the first year's working of the process.

Increased tank capacity is another advantageous feature of the process, as one can reckon on, at least, a 30% to 33% increase, and there is not the least doubt but that a large number of slimes plants along the reef would derive a great benefit from this source, even if it only gave them a 10% to 15% increase.

A considerable saving of time is effected in the washing, as from 12 to 18 hours is saved per charge. This advantage being especially appreciable during the winter months.

Regarding umber, I have little to say. We experimented with this oxidising agent for a short time, but found we obtained just as good results without using it.

In conclusion, I might say that I think it is due to the Adair-Usher process that these advantages which we have obtained on the Geldenhuis Estate should be brought before the members of this Society, more especially as one of the members, in his discussions on Mr. Adair's paper, seemed to have overlooked the good points and brought to your notice imaginary "defects," "drawbacks" and "objections" before giving it a fair and extended trial.

Mr. Geo. Melvill (*Member*): Being personally connected with the original experiments on a large scale at the Crown Reef Mine, a few notes on the working results of the Adair-Usher process will, no doubt, be of interest.

The first experiments were carried out in a tank 32 ft. diameter, 14 ft. deep, with four revolving arms at the bottom of the tank, connected on to a 9-in. down pipe. These arms were made of 3 in. pipes with 50 ½ in. holes in each pipe. Each charge consisted of about 100 tons. The results obtained were excellent, but the mechanical part of the business was defective, and consequently one could never rely on the charge being completed. This apparatus gave better results than the stationary pipes at present in use, as the solution was more evenly distributed, but the cost of maintenance was too great.

The plant at the Crown Reef mine consists of :—

5 Collecting vats	30 ft. × 10 ft. 0 in.
5 Treatment vats	32 ft. × 14 ft. 0 in.
3 Treatment vats	50 ft. × 13 ft. 0 in.
2 Filters	20 ft. × 5 ft. 6 ft.
2 Filters	24 ft. × 8 ft. 0 in.
1 Solution sump	40 ft. × 11 ft. 0 in.
3 Precipitation boxes	30 ft. × 6 ft. 0 in. × 3 ft. deep.

With this plant I am treating 8,000 tons of slimes per month, and it is capable of doing still more.

The Usher apparatus in the treatment tanks consists of a 10-in. down pipe at the centre of the tank, and projecting 6 ft. above the top of the tank, and sufficient 1 in. pipes radiating from the distributor at the bottom end of the pipe, so that the distance between the pipes at the periphery is not more than 2 ft. In each pipe ⅛ in. holes are drilled at an angle of 30° to the bottom of the tank, with centres varying from 12 in. at the centre of the tank to 3 in. at the periphery. The radiating pipes are kept about 2 in. from the bottom of the tank.

Treatment.—About 100 tons of slimes are collected in each charge, and when settled the charge is transferred to one of the treatment tanks, with solution (·02%) from the next tank to be discharged, care being taken that solution is being introduced in the centre pipe from the time the transfer begins, until the treatment is stopped. After the transfer is completed the charge is circulated for two hours or more if necessary, and washing is continued with precipitated solution until the assay value of the clear solution leaving the tank is about ·20 dwt. or less, when the charge is allowed to settle. The settled solution is then used for transferring another charge. The residue samples are taken by means of a long tin tube (2 in. diameter) with a flap valve at the bottom end. The rate of washing is regulated by the settlement of the suspended slimes and the box and filter capacity, always bearing in mind that the faster the washing the sooner will the treatment be completed.

With 100-ton charges in the smaller size tank (32 ft. × 14 ft.) washing is at the rate of 10 to 15 tons per hour, being completed in about 48 hours, and settlement about 36 hours.

It is very important that every hole in the radiating pipes is clear before transferring a charge. With the small size tanks the holes can be cleared through the centre pipe, but with a 50-ft. tank each pipe has to be flushed out independently, with a strong pressure of water through a 1-in. hose, from the periphery end of the pipe. This takes a little time, but it is time well spent.

We have been using the process on the entire slimes plant for the past twelve months, and the results have certainly been very satisfactory—more actually than theoretically.

The chief benefits derived by the process are :—

(1) Lower residues.

(2) Less dissolved gold retained in residues.

(3) Saving in power and consequently maintenance of pumps.

(4) Reduction in time of treatment, which means increase in capacity of plant.

(5) Less work for shiftmen.

From a charge of 2·2 dwt. the residues are ·3 dwt. as against ·45 dwt. before starting the process ; but I consider the greatest saving is in the gold in solution in residues sent to slimes dam.

With the decantation process the final drainings assayed from 10 gr. to 16 gr., and with the Adair-Usher process that solution assays from 2 to 4 gr.. or a saving of 10 gr. per ton of solution. Now for every 100 tons of dry slimes discharged there are about 90 tons of solution, therefore for 8,000 tons of slimes (our monthly tonnage) there are 7,200 [illegible]s of solution, which means a saving of 150 oz [illegible]e gold per month.

The pumps run only about two-thirds the time they used to, and consequently the spindles and fans do not require renewing so often, which is a great consideration.

Before installing the process the plant was too small to cope with the tonnage, but now, with one treatment tank less, there is plenty of time for treatment. The charge is completely treated in 72 hours whereas it used to take 100 hours.

Mr. White does not seem very favourably impressed with the process. He has evidently not had much practical experience with it, as there is no doubt the Adair-Usher process is a decided improvement on the old decantation method.

I would like to point out to the author that there is no supposition whatever on my part regarding the decrease of alkalinity of solution through the use of umber. It is an absolute fact, and if he will come out to the Crown Reef I shall be only too pleased to prove it to him. I did not mention in my remarks that umber was acid, and I agree with him that there is no free acid in it, but I have proved beyond doubt, on a large and small scale, that when it is mixed with an alkaline solution, a loss in alkalinity does occur. I believe Mr. Alexander, cyanide manager, Robinson Gold Mining Co., will agree with me, as he used umber on a large scale, but had to stop it to save his lime bill.

When the author is convinced that a loss of alkalinity does occur, through the use of umber, we shall look forward to an explanation.

REMINISCENCES OF THE EARLY RAND.

(Read at August Meeting, 1908).

By M. H. Coombe (Member).

DISCUSSION.

The President: I would like to say that I have brought up the old photograph of Johannesburg, as I promised to do at the last meeting. It will be passed round the room, and I will ask you to be careful with it for it is very old and very tender, having been very much mishandled during the war.

The meeting then closed.

Contributions and Correspondence.

SOME FEATURES OF SILVER ORE TREATMENT IN MEXICO.

(Read at January and March Meetings, 1908.)

By W. A. Caldecott, B.A., F.C.S. (Past President).

The following comments upon the above paper have been received from Mr. Bernard MacDonald, President of the Guanajuato Mines Selection Company:—

(1) The only innovation on the old patio process methods was the substitution of a mechanical mixer for the tramping down by horses and mules on the Torta. This was done at the Loreto patio plant of the Real del Monte, which was torn out and a cyanide plant substituted after the Boston Company purchased the property. Fine grinding in preparation of the pulp for treatment by the patio process was recognised as being essential to high extraction. The Mexicans had a saying that this was necessary because "quicksilver has no teeth." By the way, we all have found out that the cyanide solution has no teeth either. Aside from the mechanical features mentioned, there has been no recent change in the patio treatment, the chemistry remaining practically the same, and this is likely to be the case with cyanidation, *i.e.*, the important improvements will be by mechanical means.

(2) The patio process is now only worked at Pachuca, and a few isolated camps, but is being generally abandoned everywhere. Some of the old conservative companies who have been carrying an expensive patio plant on their books as an asset, do not like to destroy that asset and add the cost of a new cyanide plant; hence, for these reasons they are continuing to use the patio process, but, doubtless, when the price of silver advances, they will erect cyanide plants.

(3) The percentage of concentrates in the ores treated by the Guanajuato mills is not uniform, differing both in weight and value in each mill. In the Pastita mill, which is the only one in Guanajuato milling ore exclusively from the Veta Madre vein in depth, the concentrates contain about 50% of the values in about 2·5% of the weight of the crude ore. The Guanajuato Reduction and Mines Co. treat mostly dump ores, which were sorted out by the old miners, and the concentrate from these show lower values than from the ore taken direct from the vein. Other mines in the district vary from each other to a greater or less extent.

(4) The concentrates can be treated successfully by cyanidation if ground fine, but that would necessitate a small special plant, and the smelters are now giving favourable treatment charge for the Guanajuato ores, and this is preventing the companies from erecting special plants for the treatment of the concentrates. I wrote an article showing the results of my laboratory tests on the concentrates by cyanidation. The article appeared in the *Engineering and Mining Journal* of December 23rd and 30th, 1905. This article showed the results obtained then, and these have been verified by numerous subsequent tests.

(5) Antimony and arsenic are practically absent from the Guanajuato ores, and are considered negligible factors in the scheme of treatment.

Mr. E. M. Hamilton writes from London as follows:—

Owing to special circumstances I have only just now been able to study the June number of our

Journal for the current year. I note therein that Mr. S. H. Pearce on discussing Mr. W. A. Caldecott's paper, "Some Features of Silver Ore Treatment in Mexico," attributes to me the idea of using mercuric chloride to aid in the solution of silver from its ores during cyanidation. It is true that I was the first, as far as I am aware, to make use of it on the plant to which he refers, but I derived the idea originally from Mr. H. J. Durant, whom I found using it for this purpose at Minas Prietas, in the State of Sonora, Mexico, eight or nine years ago. In the latter instance I may state that after repeated and careful investigation I decided that there was nothing gained by its use, which was accordingly abandoned.

Queries and Replies.

Concrete Piles in Mills.—I understand concrete piles are being used in mills, on the Rand, in place of pine. Could any member supply me with the component parts, proportions, etc., adopted, and also inform me whether they are proving a success?—L.H.L. (Rhodesia).

Reply.—At the Witwatersrand Deep there are ten stamps arranged with concrete blocks under the mortar boxes. The latter are bolted to 9 in. anvils, secured to the concrete, in which the bolts run in accessible grooves. The composition of the concrete is five parts of stone, three parts of sand, and one part of Portland cement, except in the top 3 ft., in which the stone is reduced to four parts. The concrete piles answer very well and give less vibration; the above-mentioned have been in use about four years. I understand that the whole of the Village Deep mill is erected on concrete piles, whilst in the Knights Central mill, now in course of construction, the anvil block is dispensed with, a mortar box with a thicker bottom being used instead.—A.S.

Obituary.

The death is announced with much regret of Dr. William Ironside Moir, M.B., Ch.B., D.P.H. (Aberdeen), who died suddenly on September 28th at the Johannesburg Hospital. Dr. Moir, who was a brother of Dr. Jas. Moir, Vice-President of the Society, joined the Society in May, 1904.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Action of Hydrogen Sulphide on Alkaline Solutions of Zinc.—"Zinc sulphide or zinc hydrosulphide precipitated from alkaline solutions by means of hydrogen sulphide is dissolved again if the action of the gas is continued for some time. Hence, in the separation of manganese from zinc by adding an excess of alkali hydroxide, filtering off the manganous hydroxide, and precipitating the zinc in the filtrate by means of hydrogen sulphide, the whole of the zinc sulphide may be redissolved in 15 to 20 minutes, by a rapid current of the gas, and may conceivably be overlooked."—L. W. McCay, *Journal of the American Chemical Society*, 1908, *30*, 376—378. —*Journal of the Society of Chemical Industry*, April 15, 1908, p. 334. (A. W.)

Sulphides and Double Sulphides. — "The mechanism of the technical preparation of cinnabar by the wet process is explained, as follows: When black mercuric sulphide is digested with an alkali sulphide or polysulphide solution, a certain quantity of the double sulphide, $5HgS,K_2S,5H_2O$, is formed. As the temperature gradually rises, this is partially decomposed with separation of red crystalline mercuric sulphide. On now cooling, a further quantity of double sulphide is formed from the amorphous mercuric sulphide, and this again decomposes as the temperature rises. By treating excess of black mercuric sulphide with excess of a dilute solution of potassium sulphide, and allowing to stand in a closed vessel at the ordinary temperature, the variations in temperature during day and night suffice to gradually effect the solution of the whole of the mercuric sulphide and its separation in the crystalline form by means of the reactions just described. It is generally stated that stannous sulphide is soluble in alkali sulphides, and this is utilised for its separation from insoluble sulphides. Actually both the solution and the residue contain tin. In order to avoid error in such separations, it is best to use ammonium polysulphide instead of ammonium sulphide."—A. Ditte, *Ann. Chim. Phys.*, 1907, [8], *12*, 229—277. *Chem. Zentr.*, 1908, *1*, 799—804.—*Journal of the Society of Chemical Industry*, April, 15, 1908, p. 334. (A. W.)

Determination of Tungstic Acid. — "The purpose of this paper is to describe a method devised by the authors, by means of which all evaporations, etc., may be dispensed with, and the time thus occupied saved. This is accomplished by a titration of the amount of a standard ethylamine solution required to dissolve the tungstic acid.

About 1 gm. of the ore, if it is very low grade, 0·7 gm. if it is medium, and about 0·3 gm. if it contains over 40% tungstic acid, is very finely pulverised, and decomposed with 50 c.c. hydrochloric acid and 15 c.c. nitric acid in a 300 c.c. Erlenmeyer flask. In order to decompose a high grade ore completely, it must be distributed evenly over the bottom of the flask and must not be stirred. When the ore is decomposed completely 50 c.c. of hot water is added, and the tungstic acid is allowed to settle a few moments. The liquid is closely decanted through a good filter.

The tungstic acid is then washed by decantation four times with a 5% salt solution, using about 40 c.c. of this hot for each washing. If the precipitate is closely decanted each time, all free acid will be washed out, as will be shown by letting the last washings drop into a beaker containing water coloured by means of phenolphthalein and a drop of ethylamine solution, to be described later.

The filter should be rinsed between decantations by a stream of hot salt solution from a wash bottle. Salt solution instead of water is used for the washing because it prevents the tungstic acid from assuming the colloidal state and passing through the filter. When the last washing is finished, the filter is removed from the funnel and placed in the original flask containing the bulk of the tungstic acid.

An excess of standard ethylamine is then run into the flask from a burette. In running in the ethylamine the first few cubic centimetres should be allowed to run down the sides of the flask slowly to dissolve any adhering tungstic acid. Any yellow particles on the rim are wiped off with a small piece of filter paper and added to the flask. The filter paper is then broken up into bits with a stirring rod, the rod rinsed, and the flask closed with a rubber stopper. This is done to prevent evaporation of ethylamine which might be great enough to vitiate the results of the analysis. On no account is the flask to be warmed.

When all the tungstic acid is dissolved, which is recognised by the disappearance of all yellow particles, a drop or so of a neutral phenolphthalein solution is added and the excess ethylamine titrated back with oxalic acid solution. The disappearance of the pink colour is not sharp, so it is better to add a drop or two of oxalic acid in excess, so as to be sure the colour is gone, and then bring it back with a drop or so of ethylamine. In this way a sharp end point is obtained. From the known relation of the oxalic acid solution to the ethylamine solution, calculate the number of cubic centimetres of ethylamine used up in dissolving the tungstic acid. From this and the ethylamine standard, the percentage of tungstic acid in the ore is readily calculated.

Pure ethylamine in a 33⅓% solution is on the market (Kahlbaum's preparation is very satisfactory). Weigh out 6 gm. of this and make up to 1 litre.

Oxalic Acid Solution.—Weigh out 2·8 gm. of pure oxalic acid and make up to 1 litre.

Standardisation.—Find the value of 1 c.c. of oxalic acid solution in terms of cubic centimetres of ethylamine solution, using phenolphtalein as an indicator. Run into a beaker 20 c.c. oxalic acid solution. Make acid with sulphuric acid, and titrate hot with a $KMnO_4$ solution of known iron standard."—J. B. EKELEY and G. D. KENDALL, Jr.—*Mining World*, May 9, 1908, p. 760. (H. A. W.)

ANALYSIS OF LEAD SLAG.—"In sampling lead slag, the sample should always be chilled. The chilling renders the slag brittle, making it much easier to grind and dissolve. It is chilled either by pouring the molten slag into water, or by taking the sample on an iron rod and plunging it into water.

In lead smelters the slag is assayed daily for iron, lime, silica and lead. It is assayed at intervals for baryta and magnesia, if these metals are known to be present.

Lead.—Make up a lead flux composed of one part borax, two parts potassium carbonate, and three parts sodium bicarbonate. Weigh 10 grams of slag into a 10-gram crucible containing 20 grams of lead flux and 5 grams of argols. Add 1 gram of silver to act as a collector of lead.

Fuse in the muffle, beginning at a low red heat. Keep at this heat until fused, which will take about 20 minutes; then raise the heat for 10 minutes, and pour. Weigh and subtract 1 gram from the weight on account of the silver added. Each 100 milligrams of the remainder will represent 1% lead.

Silica, Lime and Iron.—While the lead is fusing, the silica, lime and iron determinations may be carried forward.

Weigh 1 gram of slag into a No 2 beaker, and add about 25 c.c. of water, and stir. While the slag is in suspension add 10 c.c. of hydrochloric acid; cover the beaker with a watch glass, and place on the hot plate.

Weigh 1 gram of slag into a small evaporating dish, and add about 1 c.c. of water, and stir with a short blunt rod. Add 2 or 3 c.c. of hydrochloric acid; place on a sand bath, and stir constantly while dissolving. As the assay goes to dryness, work the silica up on to the sides of the dish, so that when the assay is dry there will be none in the centre. Allow the assay to dry, and scrape the silica down into the centre of the dish, breaking up all large pieces.

Cool and add a few cubic centimeters of water and about 5 c.c. of hydrochloric acid: cover with a watch glass, and boil. Filter and wash, first with hot water, then wash out the dish with hot dilute hydrochloric acid and pour around the edge of the filter, then wash with hot water again.

Place the precipitate in annealing cup, and burn in the muffle, and weigh as silica. This operation takes about 30 minutes. When it is finished the lead will be ready to pour.

To filtrate add a few drops of nitric acid and boil. Remove from the hot plate, and add ammonia until the iron begins to precipitate, then add a hot solution of oxalic acid until the iron precipitate is dissolved. Repeat this operation three times, finally adding an excess of oxalic acid.

Allow the precipitate to settle in a warm place, and filter. Wash five times with boiling water to remove all traces of oxalic acid. Place the precipitate in an annealing cup, burn in the muffle, and weigh as lime.

Another method in common use is to dissolve the precipitated calcium oxalate in dilute sulphuric acid. Boil and titrate with a standard solution of potassium permanganate.

The slag in the No. 2 beaker is now dissolved and is ready for treatment. Add a few drops of stannous chloride until any yellow color present disappears; then about 25 c.c. of a saturated solution of mercuric chloride, and titrate with a standard solution of potassium bichromate.

If barium is to be determined in the insoluble residue, burn and weigh in a platinum crucible. Moisten with water and add about 1 c.c. of sulphuric acid and 2 or 3 c.c. of hydrofluoric acid, and evaporate to dryness. Ignite and weigh as barium sulphate. The weight of the barium sulphate subtracted from the original weight of the insoluble residue will be the silica.

If a platinum crucible is not to be had, burn the insoluble residue in a porcelain crucible. After weighing, fuse with 2 or 3 grams of sodium bicarbonate. Place a scorifier in the muffle upside down and put the porcelain crucible on it. This will keep the bottom of the crucible clean. Allow fusion to remain in the muffle for 30 minutes to cool and dissolve the fusion in hot water, and filter and wash until the washing show no precipitate with barium chloride.

Dissolve the barium carbonate on the filter with dilute hydrochloric acid, and wash until the washings show no precipitate with sulphuric acid. Boil the filtrate and precipitate with a hot solution of sulphuric acid. Allow to settle in a warm place, filter, wash, burn, and weigh as barium sulphate.

Magnesia is determined in the filtrate from the calcium oxalate precipitate. Boil the solution, and precipitate the iron with ammonia, boil, and filter. Wash with hot water, and invert the funnel, and wash the ferric hydrate precipitate back into the beaker. Dissolve in hydrochloric acid and again precipitate with ammonia. Boil and filter through the same filter and wash with hot water.

Cool the filtrate to as low a temperature as possible, and precipitate the magnesia with sodium-ammonium phosphate. It may be necessary to stir quite violently. When the magnesia begins to precipitate, set the beaker aside and allow to stand over night.

Filter and wash with hot water containing ammonia. Burn in the muffle, and weigh as magnesium pyrophosphate. The weight of the precipitate multiplied by 0·36036 will give the weight of magnesium oxide."—EVANS W. BUSKETT.—*Mining World*, May 9, 1908, p. 754. (H. A. W.)

DETERMINATION OF NICKEL.—"For the separation and determination of nickel in the presence of any proportions of cobalt, iron or manganese, the following method is presented in *Comptes Rendus*, vol. 145, 1907, p. 1,334.

Treat the ore with acids until all metallic parts are in solution, precipitate the alkaline earths by adding ammonium sulphate, filter, and concentrate the filtrate. Add to it a large excess of saturated solution of ammonium molybdate, then ammonium chloride, and heat for several minutes at 80 or 90 C. Cool, preferably with ice, stirring all the time. The precipitate contains all the nickel as nickel-ammonium molybdate, and nearly all the iron; all the manganese and cobalt, and a trace of iron, remains in solution. Filter, wash with saturated ammonium chloride solution, and then put the filter and precipitate into a beaker. Treat with boiling water, add ammonium chloride, then ammonia, and filter. All the iron is precipitated while nickel remains in solution.

The nickel can then be determined electrolytically, after precipitating it with an alkali and redissolving in sulphuric acid."—*Engineering and Mining Journal*, May 2, 1908, p. 910. (G. H. S.)

THE SOLUBILITY OF POTASSIUM CHLOROPLATINATE.—In a series of observations carried out by the authors the values found were considerably lower than those of previous investigators. Those found now along with the values given by Bunsen are shown below:—

Temperature C.	Authors.	Bunsen.
0	0·4784	0·74
10	0·5992	0·90
20	0·7742	1·12
30	1·000	1·41
40	1·355	1·76
50	1·865	2·17
60	2·444	2·64
70	3·055	3·19
80	3·711	3·79
90	4·360	4·45
100	5·03	5·18

The author's values give a continuous curve. The solubility in methyl and ethyl alcohols was also determined, and it was found that the solubility of the salt in alcohol water solutions varies in a regular manner with the amount of alcohol present, decreasing gradually as the percentage of alcohol increases. In potassium chloride solutions the solubility of the chloroplatinate decreases with the increase in concentration of the potassium chloride until a concentration of 1 gm. molecule per litre is reached. Beyond this point the increase in concentration of the potassium chloride has practically no effect. The solubility in NaCl solutions increases rapidly until a concentration of ·05 gm. molecules per litre is reached. For more concentrated solutions the increase in solubility is small and almost proportional to the increase in concentration of the sodium chloride.—G. S. JAMIESON, L. H. LEVY and H. L. WELLS.—*Journal of the American Chemical Society*, xxx., 5, May, 1908, p. 747. (J. A. W.)

METALLURGY.

BOILING POINTS OF METALS.—"A paper by Dr. Oliver P. Watts dealt with 'the metals in order of their boiling points as arranged from Moissan's experiments in the distillation of metals and alloys.

Dr. Watts gives a concise summary of the various determinations made by Moissan, and finally presents his attempt to arrange the entire series of metals treated in order of their boiling points. This is a difficult task to accomplish satisfactorily, and probably an impossible one to carry out with complete accuracy. In several cases the difference between the weights of two metals distilled is within the limit of experimental error. The specific heats and unknown latent heats of vapourisation may also cause errors in the tabulation, as the writer could not make allowance for these factors. The only known points in the series are the boiling points of zinc, 940° C., and copper, 2,100° C., as determined by Féry. The boiling point of tungsten was then assumed to be 3,700 C., and the other metals were arranged between these limits.

The number standing opposite each metal is not claimed to be its boiling point, but only as an indication of its relative position in the series which begins with zinc at 940° C., and ends with tungsten at some unknown temperature above the melting point (3,200° C.).

	C.		°C.
Zinc	940	Titanium ...	2,700
Cadmium ...	1,025	Rhodium ...	2,750
Lead	1,250	Ruthenium ...	2,780
Silver	1,850	Gold	2,800
Copper	2,100	Palladium ...	2,820
Tin	2,170	Iridium... ...	2,850
Manganese ..	2,200	Osmium ...	2,950
Nickel	2,450	Uranium ...	3,100
Chromium ...	2,500	Molybdenum ...	3,350
Iron	2,600	Tungsten ...	3,700
Platinum ...	2,650		

—O. P. WATTS.—*Electro-Chemical and Metallurgical Industry*, November, 1907. (J. A. W.)

SLIMES FILTER.—"A number of flat filter boxes, or trays, are arranged in a circle around a reaction-wheel which forms a central rotating distributor. The pulp to be filtered is fed into a hopper at the top of the wheel, which, as it rotates fills tray after tray with pulp. Vacuum is applied to the interior of the trays and draws off the liquid, and when a sufficiently thick cake has formed, the supply of pulp is stopped and wash-liquor is delivered on to the surface of the cake from another arm of the central rotating distributor. When washing is finished, the supply of wash-water is stopped, and the cake is drained dry by the vacuum, after which the vacuum is shut off and the filter tray reversed. Compressed air is then admitted and the cake of residue is blown off the filter into a receptacle placed below. All these operations follow one another automatically, the duration of each being regulated by the speed at which the distributor rotates. The shutting and opening of the vacuum and compressed air valves and the reversing of the filter trays are performed by levers attached to tappets which are operated by arms projecting from the central distributor and coming into action as it rotates."—H. D. Fitzpatrick, Glasgow. From H. M. Leslie, Marrikuppam, India, Eng. Pat., 13.569, June 12, 1907.—*Journal of the Society of Chemical Industry*, March 16, 1908, p. 214. (W. A. C.)

Chloride of Barium for Hardening.—"While the lead bath can be used for heating high-speed tools, and may be very useful in the case of small ones, it does not prevent entirely the trouble of oxidation. None of the other metals available for metallic baths are any more suitable.

A partial solution of the problem was found when barium chloride was tried. The chloride melts at a temperature somewhat below that best adapted to hardening high-speed steel, and can readily be raised to the proper temperature in a suitably designed furnace. A piece of steel when heated in a bath of barium chloride and then withdrawn is coated with a thin film which effectually preserves it from contact with the air until it is plunged into the cooling bath or cooled in the air blast.

The process has recently been taken up and developed by the Firth-Sterling Steel Co., U.S.A.

As now used it is exceedingly simple. Commercial barium chloride, which sells for about 1½d. a pound, is melted in a graphite crucible and brought up to a temperature of about 2,150° F. in a gas furnace. A small proportion of sodium carbonate (soda ash) is added in order to prevent fuming, and the bath is ready for use."—*Engineering Review*, June, 1908, p. 387. (A. R.)

MINING.

Rise of Earth Temperature.—"M. Durnerin has recently communicated to the Société de l'Industrie Minérale the result of his observations on temperature taken during some deep boring operations in Meurthe-et-Moselle. The principal difficulties in the observation of the earth's temperature are currents of underground water, the admission of outside water into the borehole, and the heat produced by oxidation which goes on when the hole passes through carboniferous or pyritiferous strata. He found that 1° of temperature was gained every 53 metres in Triassic sandstones, and every 16·5 to 20 metres in carboniferous conglomerates having a shaly matrix. These results confirm the previous knowledge that the rise in temperature is more rapid in the older strata. It is probable that in the future development of the deep coal measures of Lorraine, relatively high temperatures, necessitating, perhaps, the use of artificial cooling apparatus, will be encountered."—*Science and Art of Mining*, June 27, 1908, p. 529. (A. R.)

Safety Appliances in Winding.—"How to prevent damage and loss of life in coal-winding accidents occupied a good deal of attention at a recent meeting of the Manchester Geological and Mining Society. Mr. Leonard R. Fletcher (Messrs. Fletcher, Burrows and Co.) described the patent keps in use under the cages at Chanters Pit, Atherton collieries. The chief advantages claimed for these keps are, he said, (1) allowing the withdrawal of the keps from under the cage with the full load of the cage upon them, so that the engineman has no need to reverse his engines for banking purposes; (2) consequent saving of labour to the engineman in manipulating the winding engines; (3) consequent saving of time and steam; and (4) reduction of wear and tear on the winding appliances. 'After three years' experience with a set of these keps at this pit,' Mr. Fletcher added, 'I can confidently say that these claims are justified in actual practice. Since these patent keps have been put in at Chanters No. 1 Pit three-deck cages have been substituted for the two-deck cages formerly in use, and the advantage gained by the use of these keps has been most pronounced.' Mr. T. H. Wordsworth, of the New Moss colliery, followed with a description of Lee's patent safety appliance for cages. A model of the appliance was exhibited. The purpose of the inventor (one of the workers at the colliery) is to hold the cage in the shaft if the winding rope breaks where wire conductors are used. Although, Mr. Wordsworth said, it has been argued that safety appliances of this description were better left unapplied, he thought it would be quite worth the expense of applying a catch of this sort if it only operated once in a lifetime, as the upkeep was infinitesimal.—Mr. Joseph Hindley and Mr. John Stoney afterwards exhibited another patent safety appliance for use in winding. In this case it was claimed that the appliance would hold an ascending cage in the place where it happened to be when the rope broke. If the cage were descending at a rapid rate at the time of the rope breaking the arrest of the falling cage would be gradual."—*Science and Art of Mining*, June 27, 1908, p. 530. (A. R.)

Use of Explosives. — "Mr. A. H. Merrin, M.C.E., Chief Inspector of Mines, in his report to the Victorian Mines Department, gives the following to secure effective results with nitro glycerine compounds, preventing poisonous fumes and missfires:—

1. Use an efficient cap. Select for the particular kind of explosive, the cap (detonator) with the proper charge of fulminate—*i e.*, the explosive compound contained in the cap. The grade of cap to be on the strong rather than on the weak side. The longest charge requires the strongest cap; a weak cap will leave hole unbottomed. The cap to be freed from sawdust by gentle tapping. Used under water, the upper end of cap where it joins the fuse to be made water-tight (grease, pitch, clay, etc.). The cap deteriorates by exposure to damp air.

2. Select the right fuse for the kind of work. The fuse to be cut clean across and not slanting. The powder not to be shaken out of end of fuse. Fuse to be inserted in cap until it reaches the fulminate. The upper part of cap to be crimped with a broad-faced tool and not grooved so as to choke the fire in the fuse. With fuse too ragged or too large to enter cap, the end to be swaged to proper size by broad crimper. The fuse should not be kept in a damp place.

3. Do not bury cap in primer, so as to cause the burning of latter. The cap to be pushed into primer, the fuse not touching the explosive, and securely tied in that position. The primer to be pushed home with a wooden rammer into contact with charge.

4. Allow no break in the contact of plugs, due to presence of borings, or to careless charging. The plugs to be pressed home firmly with wooden rammer to fill up spaces round charge and so completely fill the hole the proper depth,

5. Tamp the charge sufficiently with clay, dry sand, or borings. Tamping not to contain any sharp particles that would damage the fuse.

6. Do not charge the hole with plugs damaged by storage in a damp atmosphere. The explosive to be stored in a dry and cool place.

7. Do not charge the hole with frozen or partially frozen plugs. The plugs to be soft and plastic. A stronger cap to be used under cold conditions.

8. Do not overcharge the hole."—*The Australian Mining Standard*, July 15, 1908, p. 64. (W. A. C.)

Dust in Coal-Mining.—"In connection with the important matter of dust in coal mines, says *Engineering*, Mr. B. F. Jones, of the Westmoreland Coal Co., Irwin, Pa., U.S.A., has recently supplied figures of tests made at their mines to *Mines and Minerals*. Compressed air machines were used, and the results of cutting with the hammer-type drill, with chain-machine, and by hand were compared. All the cuttings were passed over a No. 16 screen having 256 meshes per sq. in., and then over a No. 40 screen of 1,600 meshes per sq. in. Reduced to the same basis the hammer-type drill produced cuttings to the amount of 10·95% of the coal mined, the chain-cutter 5·86%, while hand-mining produced cuttings to the amount of 14·45% of the coal mined. The cuttings and dust that passed through a No. 16 screen amounted to 2·4% of the coal mined in the case of the hammer drill, 1·06% for the chain machine, and 1·11% in hand-mining. The dust passing through the No. 40 screen amounted for the hammer-drill, to 1·25% of the coal mined, 0·494% for the chain machine, and 0·408% by hand-mining. According to these results, therefore, the pick chain-cutting machine makes practically as little dust as hand-working."—*Indian Engineering*, July 4, 1908, p. 13. (A. R.)

MISCELLANEOUS.

Moisture, and the Strength of Wood.—"Very little being known about the influence of moisture on the strength of wood, the United States Forest Service made some time ago a thorough study of the question. The results of its investigations, observes the *Engineering Times*, are interesting and instructive. It has been found that the relation of moisture to strength follows a definite law. The strength of all kinds of wood increases rapidly with proper drying, the amount of increase depending on the species and the degree of dryness. Thus the strength of a piece of unseasoned red spruce may be increased over 400% by a thorough drying at the temperature of boiling water. But the strength decreases again as the wood reabsorbs moisture. Air-dried wood protected from the weather, and containing 12% of moisture is, according to species, 1·7 to 2·4 times stronger than when green. Drying also increases the stiffness of wood. These conclusions have been drawn from pieces of small cross-section, not exceeding 4 in. × 4 in. Large timber requires years of drying before the moisture is reduced to the point at which the strength begins to increase. It has been found that, under normal conditions, wood fibre will absorb a definite amount of moisture. Additional water only fills the pores. It has also been found that the water which simply fills the pores has no effect on the strength. The fibre saturation point is :—For long leaf pine 25, red spruce 31, chestnut 25, red gum 25, red fir 23, white ash 20·5; Norway pine 30%, estimated on the dry weight of the wood. Timber that has been dried and resoaked is slightly weaker than when green."—*Indian Engineering*, March 14, 1908, p. 175. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

The Electric Furnace, its Evolution, Theory and Practice. By Alfred Stansfield, D.Sc., A.R.S.M., Professor of Metallurgy in the McGill University, Montreal. 53 illustrations. Price, 2 dolls. (Toronto, Canada: *The Canadian Engineer.*)

"Although a great deal has been written recently about the different types of electric furnaces and their industrial application, the information available is scattered through technical periodicals published all over the world, and no book giving the gradual development, from the first experimental furnace constructed by the late Sir William Siemens in 1878 to the present modern furnace now employed on an industrial scale, has been available, either for the student or for the technical man interested in the subject. It is particularly fitting that the task of summarising all the most recent information, and the discussion of the future possibilities of the elctric furnace, should have been undertaken by the Professor of Metallurgy of McGill University, as Canada, with its vast undeveloped waterfalls, has very special reasons for being exceptionally interested in the economic application of electrical energy to industry. The book, after a short history of the subject in the first three chapters, gives a very clear description of the various types of furnaces at present in use, and discusses the efficiency of the different types, and of electric furnaces generally, as compared with other furnaces. The design, construction, and operation of the furnaces are described, and one chapter is devoted to the production of iron and steel. In this chapter there is a very good general description of the principal furnaces actually in use for the production of steel, together with the most recent results obtained with experimental furnaces in Canada and elsewhere. Chapter VI. deals with the utilisation of the electric furnace for the production of alloys and general metallurgical work, as well as the various other purposes for which it has been employed on an industrial scale; and the final chapter is devoted to the discussion of the probable industrial development of the electric furnace in the near future. The book, which, though it contains only 195 pages, is well illustrated, should be very useful to the technical man or student, who, without having the time to go deeply into the subject, wishes to obtain a general knowledge of the present position of the electric furnace and of the extent to which it is being employed in our various industries. It is a very clear, concise summary and digest of what has been done, and what is at present being done, on a commercial scale with various types of electric furnaces."—*The Mining Journal*, July 25, 1908, p. 117. (W. A. C.)

Electro - Metallurgy. By John B. C. Kershaw. Price, 6s. (London: A. Constable & Co., Limited.)

"The author's aim in this book has been to give a brief and clear account of the industrial developments of electro-metallurgy in language that can be understood by those whose acquaintance with either chemical or electrical science may be but slight, and there can be no doubt that he has been eminently successful. We have very great pleasure in cordially recommending Mr. Kershaw's work to all those who wish to be *au courant* with the general scope of electro-metallurgical practice, and especially with its latest applications. The book presents in a compact form the essential features of a most important department of technology, and will be found to be a most valuable addition to the library of all who are interested in any way in applied science. The book deals with the electrolytic refining of gold, silver, and copper, and with the refining and extraction of lead, nickel, and zinc; with the electrolytic tin stripping processes; with the electric smelting of aluminium; and, as a welcome addition to existing literature, with the production of calcium carbide and cyanamide, carborundum, graphite, quartz glass, ferro-alloys, and iron and steel, by the application of the electric current. The arrangement of the book is all that could be desired, and the illustrations are excellent."—*The Mining Journal*, July 25, 1908, p. 117. (W. A. C.)

REPORT BOOK FOR MINING ENGINEERS. By A. G. CHARLTON. Second edition, revised and enlarged. Price, 7s 6d. net. (Published by Whitehead, Morris & Co.)

This useful pocket-book, the outcome of 27 years' experience in mine inspection and mine management in various parts of the world, has been compiled with the object of supplying a systematic method upon which to base reports and preserve a record in a condensed form for speedy reference. The book is arranged in skeleton form, blank spaces being left under comprehensive headings and against numerous searching questions to be filled in in accordance with the results of investigations. The information thus asked for covers the field of mining and reduction in such a thorough manner that nothing of any importance can well be overlooked. There are many useful tables and data included, and altogether the work will be found of great value to any mining engineer, who may be called upon to furnish a report on a property. (A. R.)

PEAT AND LIGNITE, THEIR MANUFACTURE AND USES IN EUROPE. By E. NYSTROM, M.E. Canada, Department of Mines, E. Haanel, Director.

This is the latest adition to the excellent series of monographs on special subjects, which is being published by the Canadian Mines Department under the energetic administration of its Director, Dr. Haanel, and readers may be reminded of the reviews which have already appeared in these columns of two published a year or two ago on mica and asbestos. The present volume attains to the high standard of its predecessors, and is practically an exhaustive treatise on the subject with which it deals. The illustrations are especially to be commended, and add considerably to the value of the book. Mr. Nystrom is to be highly congratulated on having so faithfully fulfilled his mission, and the book will be invaluable to those who are engaged in the manufacture and utilisation of this fuel. (J. A. W.)

DESIGN AND EQUIPMENT OF SMALL CHEMICAL LABORATORIES. By R. K. MEADE. 10s. nett. (Published by Chem. Engineering Publishing Co., Chicago.)

The scope of this work is amply and succinctly covered by the title. To the chemist anxious to bring his laboratory up to date, the work is valuable in that modern methods and apparatus alone are dealt with. The fixing up, arrangement and selection of working apparatus for a small scale working laboratory are clearly detailed, and it is needless to say, the assay department is not forgotten. One notes with pleasure the latest designs in crushers, fine grinders, mechanically operated agate pestle and mortar, smelting and muffle furnaces, and cupel making machines. His opening sentence might well be followed with advantage by all engaged in chemical work on these fields. It reads—"Where space permits, have a separate table for carbon combustions, for ignitions, extractions, evaporations, for titrations, precipitation with H_2S, distillations and for electro-chemical analysis, filtrations and various and sundry special tests." A well-written, readable and clearly illustrated book, worthy the perusal of all chemists and assayers. (A. McA. J.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(*N.B.—In this list* (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.*)

(P.) 333/08. William Griffith Williams. Improvements in means for heating, melting or volatilising. 18.8.08.

(P.) 334/08. George Smith. Improvements in apparatus for disposing of sand or such like material on mine dumps or other depositing sites. 18.8.08.

(P.) 335/08. Henry Makins Tait. Improvements in safety catch devices. 18.8.08.

(P.) 336/08. Robert Hutchinson Anderson. Improvements in rock drills. 20.8.08.

(C.) 337/08. Paul Sabatier. Improvements in and relating to manufacture of methane or of mixtures of methane and hydrogen. 21.8.08.

(C.) 339/08. Charles Christiansen. Improvements relating to hammer rock drills. 21.8.08.

(P.) 340/08. Allan McGregor Ritchie (1), David Cunningham McIlleron (2). An improved pole or post used for fencing or the like. 21.8.08.

(C.) 341/08. Samuel Victor Man (1), Adolphus Pillman (2). Improvements in means for purifying air. 21.8.08.

(P.) 343/08. Robert Hunter (1), Robert Leslie Polson (2). Improvements in acetylene gas generators and lamps. 24.8.08.

(C.) 344/08. Henry Samuel Potter (1), Frederick Dwight Johnson (2). Improvements in or relating to fluid pressure operated rock drilling apparatus and the like. 24.8.08.

(P.) 345/08. Charles Ward Hammerton (1), Thomas Hugh McDonald (2), Frank Porson Neve (3). Improvements in and relating to electrical signalling systems for mines. 26.8.08.

(P.) 346/08. Charles Hansen. Improvements in safety brake arrangements for mine cages and other hoisting apparatus. 27.8.08.

(P.) 347/08. James Grant Gibson. Improvements in linings for tube mills. 27.8.08.

(P.) 349/08. James Snodgrass. Improvements in the purification of water. 29.8.08.

(C.) 350/08. Walter Neal. Improvements in pebble retaining outlet for tube mills. 31.8.08.

(P.) 351/08. Herbert Lewis Lezard. Improvements in spanners. 1.9.08.

(P.) 352/08. Frederick Thomas Watt. Improvements in machines for crushing or disintegrating ore and other substances. 1.9.08.

(P.) 353/08. Carl Frederick Siegert. Improvements relating to the treatment of auriferous black sand concentrates and the like for recovery of the precious metals. 1.9.08.

(P.) 354/08. John Holgate. Improvements in reciprocating engines. 2.9.08.

(P.) 355/08. William Joseph Holmes. Improved portable ground surface pulveriser and detachable teeth for same. 2.9.08.

(C.) 356/08. Frederick Adamson. Improved diamond washer. 3.9.08.

(P.) 358/08. Frederick Victor William Swanton (1), Thomas Arnott Fletcher (2). Improved apparatus to produce power by the combined action of certain fluids of dissimilar elasticity and weight upon same. 3.9.08.

(P.) 360/08. John Francis Weedon. A new or improved reinforced concrete sleeper for railways and tramways. 7.9.08.

(P.) 361/08. James Catford Koller. Improvements in the extraction of metals from their ores and apparatus therefor. 8.9.08.

(P.) 362/08. William Thomas. Improvements in means for fixing tappets to the stems of battery stamps, also applicable for fixing collars or bosses to shafts and the like. 8.9.08.

(P.) 363/08. Henry Alford Walker (1), George Walter Compton (2). Improvements in the manufacture of earthenware pipes. 8.9.08.

(P.) 364/08. William Arthur Caldecott. Improvements in or relating to means for separating liquid from crushed ore products. 9.9.08.

(P.) 365/08. James Ralph. Improvements in rollers for supporting haulage ropes and the like. 9.9.08.

(P.) 366/08. Anastisis Marousis. Improved liquid elevating device. 9.9.08.

(P.) 367/08. Louis Thomas Dechow. Improvements in slime filtering devices. 9.9.08.

(P.) 368/08. Herman Georg Spengel. Improvements in rock boring machines. 9.9.08.

(P.) 369/08. Ernest Albert Brinkmann. Tube mill liner. 10.9.08.

(C.) 370/08. Matthew Yarrow. Improvements in joints of pipes or mains. 11.9.08.

(C.) 371/08. Albert Hibbs Taylor. Improvements in hammer drills. 11.9.08.

(C.) 372/08. Albert Hibbs Taylor. Improvements in hammer drills. 11.9.08.

(C.) 373/08. Albert Hibbs Taylor. Improvements in hammer drills. 11.9.08.

(C.) 374/08. William Prellwitz. Improvements in unloaders for fluid compressors. 11.9.08.

(C.) 375/08. William Prellwitz. Improvements in unloaders for compound fluid compressors. 11.9.08.

(C.) 376/08. William Prellwitz. Improvements in brace attachments for fluid operated tools. 11.9.08.

(P.) 377/08. Alexander Purser. A new and improved machine for drilling rock and the like. 11.9.08.

(P.) 378/08. James Jameson. Improvements in means for connecting trucks or other vehicles to a hauling rope. 12.9.08.

(P.) 379/08. Alfred Roach. Improvements in smelting, assay, and like furnaces. 16.9.08.

(P.) 380/08. Arthur William Davis. Device for concentrating and recovering metals from alluvial wash and the like. 16.9.08.

(C.) 381/08. Charles Tingley Carnahan (1), Jeremiah Murphy (2). Water attachments for pneumatic hammers. 18.9.08.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

Andrews, A. A., *l/o* Barberton; Forbes' Reef, Swaziland.

Atkin, A. J. R., *l/o* Steynsdorp: Molyneux Mine, Kraal Station.

Bawden, E. R., *l/o* England; Lilongwe Boma, Central Angoniland, British Nyassaland, via Chinde, East Africa.

Burt, W., *l/o* Johannesburg: Wanderer Mine, Selukwe, Rhodesia.

Coaton, A. A., *l/o* Germiston; Witwatersrand Deep, Ltd., P. O. Box 5, Knights.

Coe, W. C., *l/o* Wales; Ayrshire G. M. Co., Ltd., Lomagundi, S. Rhodesia.

Collings, B. L., *l/o* Salisbury, Rhodesia; Holnicote, Heavitree Road, Exeter, England.

Coulter, W. M., *l/o* Cleveland; Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg.

Dennehy, M. J., *to* corner of Alexander Road and Biccard Street, Turffontein.

Dures, R., *l/o* Langlaagte: Wanderer G. M., Ltd., Selukwe, Rhodesia.

Edwards, A. L., *l/o* Johannesburg; P. O. Box 74, Barberton.

Graham, W. H., *l/o* England; P. O. Box 5281, Johannesburg.

Hellmann, Fred., *to* 832-3, Salisbury House, London. E.C.

Jane, W. H., *l/o* Nigel; P. O. Box 178, Germiston.

Jones, J. A., *l/o* Johannesburg: Cinderella Deep G. M. Co., Ltd., P. O. Box 75, Boksburg.

Lawrie, E., *l/o* Johannesburg: No. 5 Mess, Oorgaum, Kolar Gold Fields, India.

Leyson, W., *l/o* Barberton: P. O. Box 3, Florida.

Macfarlane, T. M., *l/o* French Guinea; c/o Messrs. H. S. King & Co., 65, Cornhill, London, E.C.

Macfarlane, T. J. M., *l/o* Victor, O.R.C.: Rand Club: P. O. Box 1032, Johannesburg.

Michie, J., *l/o* Johannesburg; P. O. Shangani, Rhodesia.

Mitchell, A., *to* Sultana Reef, Surprise Siding, Rhodesia.

Moncrieff, R. M., *l/o* Korea; 21, Kensington Park Gardens, London, W.

Nicholls, R., *l/o* Krugersdorp: P. O. Box 220, Bulawayo.

Parfitt, W., *l/o* Nigel; Cinderella Deep G. M. Co., Ltd., P. O. Box 75, Boksburg.

Robinson, L. J., *l/o* Cleveland; P. O. Box 108, Germiston.

Wilkinson, Hy., *l/o* Benoni: West Rand Consolidated Mines, Ltd., P. O. Box 38, Krugersdorp.

Wisdom, G. E., *l/o* Queque; Jumbo G. M. Co., via Salisbury.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. IX. OCTOBER, 1908. No. 4.

Proceedings
AT
Ordinary General Meeting, October 17, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, October 17th, Mr. R. G. Bevington (President), in the chair. There were also present :—

36 Members : Messrs. A. McA. Johnston, E. H. Croghan, A. Richardson, Prof. G. H. Stanley, J. E. Thomas, H A. White, Prof. J. A. Wilkinson, W. A. Caldecott, W. Cullen, Prof. J. Yates, H. D. Bell, F. T. Chapman, A. A. Coaton, J. Coats, M. H. Coombe, W. M. Coulter, G. A. Darling, L. Evans, J. H. Hamill, T. Johnson, J. Kennedy, M. Knight, J. Lea, J. P. McKeown, J. T. Mitchell, T. T. Nichol, D. W. Perkin, J. F. Pyles, O. D. Ross, H. Scarf, R. Stokes, J. W. Taylor, A. Thomas, J. P. Ward and E. M. Weston.

8 Associates : Messrs. J. Chilton, J. Cronin, W. H. Graham, H. G. Kirkland, W. A. C. Tayler, A. M. Thomas, W. Waters and J. Whitehouse.

12 Visitors.

The President : Mr. Cullen has kindly undertaken to act as Secretary to-night for Mr. Rowland, who is prevented by family matters from being present.

The minutes of the previous monthly meeting, as published in the *Journal*, were confirmed.

NEW MEMBERS.

Messrs. T. T. Nichol and A. A. Coaton were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows :—

Beatty, George Hewitt, New Goch G. M., Ltd , P. O. Box 1096, Johannesburg. Mine Surveyor.
Foran, Jack Henry, M.M. Redjang Lebong, Benkoelen, Sumatra. Assayer. *(Transfer from Associate Roll.)*
Macdonald, Bernard, Apartado 33, Guanajuato, Mexico. Mining Engineer.
Mossop, William Clement, Wolhuter G. M. Ltd., P. O. Box 1160, Johannesburg. Mining Engineer.
Semple, William Boyd, Luipaardsvlei Estate and G. M. Co., Ltd., P. O. Box 53, Krugersdorp. Chemist and Assayer. *(Transfer from Associate Roll.)*

The following gentlemen have been admitted as Associates by the Council since the last meeting :—

Aspland, Christopher Hatton, Simmer Deep, Ltd., P. O. Box 178, Germiston. Mine Surveyor.
Brading, Thomas, Van Ryn G. M. Co., Ltd., P. O. Box 22, Benoni. Cyanider.
Dowling, Douglas, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Cyanider.
Halford, James Edward, Nigel Deep G. M. Co., Ltd., P. O. Box 50, Nigel. Amalgamator.

VISIT TO DYNAMITE FACTORY.

The President detailed the arrangements for the visit to the Dynamite Factory on the 24th inst.

Mr. Cullen stated that programmes would be posted to those participating in this visit, during the coming week and that visitors would be divided up into parties on the train. He proposed, during the afternoon, to give a resumé of a paper to be read before the Society at its next meeting, and which he thought would prove of great interest. He hoped they would have a very enjoyable visit.

Prof. G. H. Stanley then read the following paper :—

THE PATIO PROCESS.

By C. Perez Duarte.

The method known as "Mexican Amalgamation" or the "Patio Process" was first used by Bartholomé de Medina in the year 1557 at the "Hacienda de la Purisima" in Pachuca, State of Hidalgo, Mexico. During its period of usefulness various competent metallurgists, both Mexican and foreign, have entered into the study of the process with the idea of overcoming its disadvantages (excessive consumption of quicksilver, long time of treatment, etc.), but apart from unimportant modifications the system as at present practised is identical with that introduced by its originator.

The practice is more or less as follows:—The ore, reduced to a slime by various systems of crushing, is stored in specially prepared places called "lameros." These floors or bins serve to collect the amount of slime necessary to form a "torta" or cake, and at the same time permit by drainage and evaporation the escape of the excess of water added during crushing. In general, the slime as formed into the "torta" carries about 33% moisture. When a sufficient amount of slime of the proper consistency has been collected it is moved to the "patio," or treatment floor, by means of a "camón," which is a sort of scraper with a horse or mule for motive power and a peon for its guiding intelligence.

The "patio" is a large floor having a slight slope to allow for drainage, on which the "tortas" are formed and treated. It is generally tiled, but sometimes paved with blocks of wood. In any case the pavement must be well laid in order to reduce as much as possible the loss due to amalgam and quicksilver draining through. That particular part of the floor on which the "torta" is formed is enclosed by means of wood, the joints being made tight with clay. It is called a "cajete." Its dimensions are such that the "torta" (usually about 150 metric tons) may fill it evenly to a depth of 20 to 25 cm. (8 to 10 in.). The slime is allowed to stand in the "cajete" for from four to six days during which time, weather permitting (for the wet season causes great inconvenience), a part of its water escapes by evaporation and drainage. At the end of this time the first step in the actual treatment is given by "salting"* the torta. The amount of salt added varies according to the richness of the ore under treatment but is about 40 to 50 times the weight of silver present. The amount also varies with the character of the ore and according to whether the minerals present are inert or retard the reactions. After salting, the first "repaso" or mixing, of eight hours' duration is given. This is the mixing or agitation of the slime by horses or mules, which are made to walk about therein in order to thoroughly incorporate the salt with the slime. In the early days of the process this disagreeable work was performed by men instead of horses. Lately mechanical mixers were tried at the Hacienda de Loreto in Pachuca, and also at the Hacienda de Pastita in Guanajuato (both of which are now cyanide plants), but the results were not up to expectations.

The day following the salting, "magistral," or sometimes copper sulphate and quicksilver, are uniformly scattered over the "torta" and thoroughly mixed by eight hours of the "repaso." The mixing is repeated every third day, and after each mixing the "torta" is turned over by shovelling. The time of treatment varies from a minimum of 20 days to a maximum of 35 days. In order to add the quicksilver in a finely divided form it is placed in canvas sacks which on being compressed shower out the quicksilver in minute globules. The quantity used should be about seven or eight times the silver contents of the "torta." "Magistral" is the result of a calcination of chalcopyrite, by which a part of the copper and iron sulphides are changed into sulphates. Understanding that copper sulphate is the active reagent in the process, it will be seen that the amount of "magistral" added is a very variable figure. Generally a good grade of "magistral" carries about 30% of copper sulphate. Such quantity should be used as will give 2 to 2·5 parts of copper sulphate to 1 part of silver. Generally less than the total amount of quicksilver necessary is added the first day, and more is added later, as shown to be necessary by the "tentadura" or test. This addition is called "cebar" or to fatten the "torta." The "tentadura" consists of panning an average sample of the "torta," and observing the appearance of the mineral and the amalgam. Some experience and judgment must be called into play at this point. A test is made before and after each "repaso."

The accidents most likely to happen during the treatment are, that the "torta" will become "caliente" (hot) or "fria" (cold). The heating of the "torta" is produced by an excess of copper sulphate and results in an excessive consumption of mercury, changing it into a protochloride, which is without effect on the silver. To remedy this state of affairs more slime is added or lime, ashes or "tequesquete" (a natural carbonate of soda) used. These correctives when not carefully used paralyse the reactions in the "torta," and result in a great loss of silver in the residues. Experiments conducted by Prof. Velasquez de Leon, late of the Mining School of Mexico, show that an excess of lime decomposes the silver chloride, and produces silver oxide, which is neither decomposed by, nor will amalgamate with quicksilver. Recent practice favours the use of precipitated copper, copper amalgam, or zinc amalgam in place of the above correctives, the first-named being the most commonly used.

Ing. Manuel V. Ortega successfully substituted sulphuric acid for the "magistral" at the Negocación de Proaño (Fresnillo, Zacatecas) with considerable resulting economy, but this is only applicable in particular cases. Usually the ore contains calcite or dolomite which, by causing an excessive consumption of acid, results in greater expense than when using copper sulphate.

---

* (Mex.) "enslamorar" or "salar."

The "torta" is said to be cold when it lacks copper sulphate. The remedy is obvious. When the appearance of the globules of amalgam obtained by panning shows that more quicksilver is necessary, the "torta" is fattened by the addition of a small amount. The termination of the treatment is guided by the appearance of the "tentadura," but, before discharging, it is customary to remove all amalgam from a general sample, by panning, and assay the residues. The "pella" (ball of amalgam) is also assayed and figured as a check on the assay of the residues. When everybody is satisfied that the treatment is completed the slime is sent to the "lavadero" or washer. The washer is formed of three or four circular tubs of wood or masonry, 3·5 metres × 1·75 metres (11 ft. 6 in. × 5 ft. 9 in.) communicating one with another by pipes about 18 in. above the bottom, and the last of the series discharging to the residues dump. Each tub is provided with a mechanical agitator of three or four arms. All being in readiness, the first tub is partly filled with slime, the paddles started in rapid movement and enough water added to fill the tub. Violent agitation is continued for from 30 to 45 minutes and the mass then being reduced to a thin pulp, the paddles are moved more slowly for two or three hours in order to give the amalgam a chance to settle. The communication to the next tank is then opened and more slime and water added to the first, agitation being continued as before. In turn the slime passes to the third and fourth tub and is discharged from the last. This is continued till all the slime has been washed. The washing of a 150-ton "torta" will take three or four days. During its passage from one tub to another and, above all, at its issue from the last of the series, samples of the slime are taken and panned in order to observe if any amalgam is escaping. At the completion of the washing the amalgam is found in the bottoms of the tubs mixed with grains of sand and whatever else of a high specific gravity may have been present in the "torta." Most of this is removed by panning in "bateas" over a large tank of water. The partly cleaned amalgam is then carried to the "azogueria" or clean-up room where it is dissolved in a large excess of quicksilver, thereby floating all impurities to the surface, from whence they are removed, leaving a pure fluid amalgam.

The excess of quicksilver is removed in a "manga," which consists of a conical sack made of leather in its upper portion and canvas in the lower part, with a stout iron ring about its mouth, and is suspended from the roof. The weight of the amalgam, assisted by blows on the supports causes the quicksilver to filter through the canvas, leaving the amalgam in the state of a stiff paste. The amalgam is retorted in retorts of such variety as would need a separate paper to describe them all. A record is kept of the quicksilver used during all operations, and the difference between this and the quantity recovered represents the loss, which is about 287 gm. of quicksilver for each 230 gm. of silver recovered. While extractions and costs vary more widely than cyanide extractions and costs at the present day, we may give as a rough average an extraction of 85% and a cost of $8 (Mexican money) per ton, equal to 16s. The loss by stealing was enormous, but impossible to estimate definitely. One curious method used to prevent stealing in the old times when the "lavadero" was trodden by men instead of being mechanically agitated, was to tie their hands behind their backs during the process. Even to the present time the "Haciendas de Beneficio" are surrounded by high mud walls, and all the labourers searched as they leave the patio.

Theory.—Of that which I have read concerning the chemistry of the Patio process, the "Practica del Beneficio de Minerales de Plata Auriferous Usada en el Distrito de Guanajuato" (Treatment of Gold-Silver Ores in the Guanajuato District) written by Professor Vincente Fernandez is so strongly in accord with my own observations that I cannot do better than quote largely from his work, bringing the equations given into line with modern ideas. He says, in part, as follows:—

"I am informed that Don Federico Sonneschmidt was the first to give a scientific theory of the process. He states that the salt and the "magistral" combine to form chloride of copper which chloridises the silver. The chloride of silver is then reduced to the metallic state by mercury, which then forms calomel. This theory is, in my view, excellent, but there are some who advance strong objections. These I shall endeavour to answer.

Those who oppose this theory advance two important and potent facts. Namely that it is impossible to extract calomel from a "torta" under treatment* by the use of solvents, and that the loss of quicksilver is not enough to satisfy the reaction that supposing the formation of calomel, the presence of chloride of copper would raise the loss of quicksilver to a still greater amount, as it is strongly attacked. Since this does not happen they find the theories of this illustrious man to be untenable.

---

* Later experimenters have succeeded in extracting calomel from the "tortas" of Pachuca.—C.P.D.

Let us proceed to study these reactions:

(1) Is chloride of copper formed by the sulphate of copper and the salt? If so, why is the mercury not attacked? (2) Is the silver chloridised? (3) In what form is the quicksilver lost?

(1) Is chloride of copper formed?

When two solutions, one of sulphate of copper and the other of salt, are mixed in those degrees of concentration in which they are used in the Patio process, the faint blue colour is changed immediately to green, showing the formation of chloride of copper. This is formed by double decomposition.

$$2NaCl + CuO.SO_3 = Na_2O.SO_3 + CuCl_2$$

If to this solution, freshly precipitated copper is added in quantity equal to what the solution contains, and the solution then heated, the greenish coloration disappears and a whitish precipitate forms. The precipitate cannot be the bibasic sulphate of copper because that reaction would require copper oxide

$$CuO.SO_3 + CuO = 2CuO.SO_3$$

Since the copper was added in the metallic state the reaction must be $CuCl_2 + Cu = Cu_2Cl_2$. If in place of adding precipitated copper to the solution we add metallic silver, this loses its metallic aspect and changes to a violaceous powder of the same appearance as silver chloride after exposure to the light. This is probably a subchloride of silver. It is soluble in ammonia, or in a salt solution. Messrs. Durochet and Malaguti have obtained these same results and claim that subchloride of copper (Cu_2Cl_2) and chloride of silver are formed. The rapid chloridisation of the silver would be impossible with the ordinary salt and is attributed to the formation of chloride of copper, which in turn passes to the form of a subchloride. We see then that the formation in both experiments of subchloride of copper results from the reduction of the perchloride that exists when solutions of sulphate of copper and of salt are mixed in the same proportions as in the "torta." We infer then that the same reactions proceed in the treatment of ores by the Patio process. It may be claimed that the matrix affects the reactions, but it is impossible that this has any considerable influence because the ores are mostly quartz or silicates little or not at all affected by chemicals in the wet way, and the carbonate of lime (calcspar), the double carbonate of calcium and magnesium (dolomite), the fluoride of lime (fluorspar), pyrites and oxides of iron are not always present, nor are they in quantities sufficient to take a principal and leading part in the reactions. We must, nevertheless, admit the formation of chloride of copper in the process.

Why is the mercury not attacked?

To this question we have no satisfactory answer. When to the solution of sulphate of copper and salt, which we studied above, a small amount of quicksilver is added, and agitated, two precipitates form; both white, but one pulverulent and heavy, the other flocculent and settling with difficulty. The first is blackened by, but the second dissolves in, and gives a blue colour to, ammonia. The quicksilver becomes granular and of a lead colour or sometimes black. By washing and rubbing under water the mercury may be separated from the heavy, white precipitate. The heavy precipitate is calomel, and the lighter, subchloride of copper (Cu_2Cl_2), and the quicksilver present has lost in a few hours, 6% of its weight. This may easily be proven, but the fact remains that, when they are mixed in the "torta," the quicksilver is *not* attacked. It may be, that the action of the quicksilver on the chloride of copper is diminished by the greater affinity of the silver of the ore for chlorine in the chloride of copper. Still, when quicksilver, silver and chloride of copper are mixed, this theory is not confirmed and the quicksilver *is* attacked. The experiment must be carried further. Add some quartz sand to the mixture. The quicksilver will not then be attacked and amalgamation will be effected. The action of supposedly inert bodies in the treatment has been studied. Domeyko, chemist, of Chili, in his "Treatise on Assays," of 1876, says that Durochet and Malaguti have found the action of mercury on natural silver sulphide in a ferruginous matrix greater than on that in an argillaceous matrix in the proportion of 2·74 : 1·76, and add that in this direct action of the mercury on the sulphides, we must consider in the first place the influence of the matrix, the quantity of sulphide of silver attacked in a silicious matrix being four times the amount attacked in an argillaceous matrix in equal length of time. Secondly, the presence of certain compounds, for instance iron or copper sulphide. This "reaction by presence" may explain the fact that, while in laboratory experiments the quicksilver is attacked by chloride of copper, the reaction is retarded in the "torta" by the presence of substances which do not themselves enter into the reaction. This we may deduce from the last cited experiment and also from the experiments of Domeyko.

(2) Is the silver chloridised?

We have found that when we mix solutions of salt and copper sulphate a solution is formed, which on the addition of copper produces Cu_2Cl_2; with silver, violaceous chloride of silver; and with mercury, Hg_2Cl_2. These experiments point to the necessity for the formation of chloride of copper ($CuCl_2$). Now, when to such a solution, containing salt in excess and sulphate of copper, as in the treatment; or in other words, chloride of copper and chloride and sulphate of soda, we add a small piece of ore, such as is treated, and some drops of mercury, we observe two facts. First, the ore is darkened and presents some amalgamated points. Second, the solution contains subchloride of copper (Cu_2Cl_2) partly in solution and partly in suspension, and also chloride of silver in solution.

The specimen is darkened by a coating of black sulphide of copper. This powder, after being removed from the specimen by washing in water, strongly colours ammonia blue. This reaction may explain the formation of copper sulphide:

$$\mathrm{Ag_2S + CuCl_2 = 2AgCl + CuS} \qquad (1)$$

Domeyko says that under these conditions Durochet and Malaguti have observed the formation of chloride of silver, subchloride of copper and free sulphur. I am of the opinion that not chloride but subchloride of silver is formed, according to the following reaction:

$$\mathrm{Ag_2S + CuCl_2 = Ag_2Cl + CuCl + S} \qquad (2)$$

Accepting either the hypothetical reaction (1) or the observations of these chemists, we may be sure of the fact of the chloridisation of the silver, since one supposes the reaction and the other confirms it.

The amalgamated points may be due to the following reaction:

$$\mathrm{Ag_2S + CuCl_2 + Hg = AgCl + AgHg + CuCl + S} \qquad (3)$$

or to the following:

$$\mathrm{2Ag_2S + CuCl_2 + 2Hg = 2AgCl + 2AgHg + S + CuS} \qquad (4)$$

Domeyko says positively that the subchloride of silver in contact with the salt is converted into chloride of silver, which is dissolved, and metallic silver, which separates out. The subchloride formed in the first two reactions might decompose thus, and the resultant metallic silver form the amalgamated points noted:

$$\mathrm{Ag_2Cl + NaCl + Hg = NaCl.AgCl + AgHg}$$

The liquor contains subchloride of copper resulting from the reduction of the perchloride (Reactions 2 and 3), and also contains chloride of silver, which is easily proved by diluting, when it precipitates before the less heavy chloride of copper.

If these reactions, which simulate those of the Patio process point to the chloridisation of the silver, there remains no reason to doubt its formation in the "torta."

Before passing to the third and last question, I wish to call attention to a very interesting point concerning the theory of action. Perhaps one of the strongest objections that has been advanced to the theory of the chloridisation of the silver (the theory of Sonneschmidt), is that the amount of quicksilver lost is not sufficient to satisfy the necessary reactions. We may represent this theory as follows:

$$\left.\begin{matrix}\mathrm{2NaCl}\\ \mathrm{CuO.SO_3}\end{matrix}\right\}=\left\{\begin{matrix}\mathrm{Na_2O.SO_3}\\ \left.\begin{matrix}\mathrm{CuCl_2}\\ \mathrm{Ag_2S}\end{matrix}\right\}=\left\{\begin{matrix}\mathrm{CuS}\\ \left.\begin{matrix}\mathrm{2AgCl}\\ \mathrm{n.Hg}\end{matrix}\right\}=\left\{\begin{matrix}\mathrm{Hg_2Cl_2}\\ \mathrm{Ag_2Hg^{\mathit{n}-2}}\end{matrix}\right.\end{matrix}\right.\end{matrix}\right.$$

That is, in effect, that to extract one equivalent of silver we must lose one of quicksilver. In other words, for each 8 oz. (1 marco) obtained we must pay 14·8 oz. of mercury. The actual loss in the patio is about 9 or 10 oz. per "marco" of silver, which includes a heavy mechanical loss, so this theory is certainly very far from the practice.

As the formation of chloride of copper, the chloridisation of the silver, and the reduction of the chloride to the metallic state, by means of the quicksilver cannot be disproved, some have supposed that in all our ores arriving at the patio half the silver is in the metallic state and some others that the quicksilver is lost in the form of corrosive sublimate, which requires for its formation only half the amount of quicksilver required to form calomel.

The first supposition would obviously diminish loss of mercury since it would amalgamate directly and lead to no formation of calomel.

The second supposition requires only one equivalent of mercury for two of silver, but we cannot admit it because, for example, in a "torta" containing 6 "marcos" of silver per "monton" (937 gm. per metric ton), the result would be that each "cuartillo" (456 c.c.) of water would contain 11 "granos" (·55 gm. of corrosive sublimate (·12%)*, and as neither the workmen who stand in the "torta" for hours at a time, nor the mules, which in addition eat the mud in considerable quantities (as is proved by the presence in their intestines of balls of amalgam sometimes as large as a lemon), nor the pigeons which drink the water collected in the hoof prints, suffer any ill results, it is clear that the quicksilver lost is not in the form of corrosive sublimate. Further, corrosive sublimate in the presence of free quicksilver is completely changed into calomel.

(3) In what form is the mercury lost?

In the previous discussion we have given an idea, and its proofs, that the silver is chloridised; now for its reduction we may assign various reasons—the sunlight, the iron of the shoes of the mules, and the quicksilver. We cannot attribute all the reduction to the first two because neither is all the silver chloride exposed to the sunlight, nor are mules used to tread the "torta" at all plants. (In early days the "repaso" was effected by men, who certainly did not wear horseshoes, and in the Hacienda de Pastita in Guanajuato the agitation was effected by wooden stirrers moved mechanically.) The quicksilver then must be the reducing agent. In what form is it lost? Not in the form of corrosive sublimate because we have seen that this would be present in sufficient quantity in the water of the "torta" to produce effects on men and animals which are not apparent. Further, that it could not exist in this form in the presence of an excess of mercury. Hence, it must be lost in the form of calomel.

---

* I give the weights and measures in the original with a view to showing how the English system looks to a Mexican. The old Mexican system of weights and measures was, if possible, worse than the modern English, but is now superseded by the metric except in isolated places in the interior. Nor were the old units standard between one part of the country and another. For instance, the "monton" mentioned here is the Guanajuato "monton" and equivalent to 1,473 kgm. while the Pachuca "monton" equals 1,280 kgm. The "marco" mentioned elsewhere is also an old standard and is equal to 230 gm.

The President: The thanks of the Society are due to Mr. Perez Duarte for putting this exposition of the Patio process before us. Of course, it is a process which many of us connected with metallurgy have read of in various works of reference, but I have certainly never seen it so thoroughly gone into. The paper is the more interesting on account of its having been written by a Mexican who is actively engaged on the Patio process. The process, if not dead, is dying, and therefore it is very fortunate for us to have this description by one who is working it, and thus place it on record in our *Journal*.

Mr. W. A. Caldecott (*Past-President*): Our President has already pointed out the extremely interesting nature of this paper from an historical point of view, and that it was written by a Mexican gentleman, who was educated as a metallurgist in the city of Mexico, and has been familiar with the Patio process all his life. The paper was originally written in Spanish, and translated for our *Journal* by the kindness of Mr. Walter Neal of the Dos Estrellas Company. The term "patio," or court-yard, is also of interest historically, since during former disturbed times in Mexico the owner of the mine naturally preferred to carry on his metallurgical operations in his own back-yard, surrounded by a good high wall, and thus prevent too great a discrepancy between the silver extracted from his ore, and that which he actually recovered as bullion. From a humanitarian standpoint it is a good thing that the patio process is practically defunct, inasmuch as the unfortunate mules or horses employed in treading the salt, blue-stone, and mercury pulp do not last many months.

The process was invented at Pachuca the year before Queen Elizabeth came to the throne of England, and during her reign the English seamen were so active in capturing Spanish treasure ships on the Spanish Main, that their reputation as pirates still lingers. I was informed that some of Drake's or Hawkins' men, during one of these patriotic expeditions, were captured by the Spaniards and were sent as prisoners to Pachuca, where they were employed in treading the "torta" during the brief remainder of their lives. In fact, men were used for this purpose up to the year 1793, and mechanical devices only came in of late years when the process was nearing extinction.*

As illustrating the rapid changes which have recently taken place in the treatment of silver ores, Mr. Bernard MacDonald of Guanajuato has kindly sent me a paper,† published in 1899, in which the author, Mr. Roberto Fernandez, refers in rather a deprecatory manner to certain modern processes, presumably the early trials made of the cyanide process, and says that if the Patio process were better understood, it would be more generally used. He also incidentally mentions ores carrying 1,000 gm. to 2,000 gm. of silver per metric ton as being of medium grade, but I need hardly say that with the cyanide process silver ores carrying only a fraction of these amounts are nowadays proving very profitable propositions.

THE THEORY OF BLASTING WITH HIGH EXPLOSIVES.

By E. M. Weston, A.S.M.B. (Member).

I venture again to write a paper on a subject about which I know very little, and my excuse is that I want information, and wish to see if anyone can give it.

I have been reading most of the works published on blasting to see if they could give me any data that would be useful in checking the work done in breaking rock in development faces and in both wide and narrow stopes in our mines. I think it will be found that these books have been written by engineers, who apparently have no great knowledge of underground conditions, and who deal with the subject mainly from a quarryman's or railway contractor's point of view. This is, I think, the explanation of the fact that we have rules laid down, based apparently on clearly proved mathematical deductions from known forces and resistance, which any right thinking miner breaks every day of his life for obvious economic reasons. Students of this subject would do well to remember that whole discussions and theses of these books are set out with the object of showing how to break the rock with the smallest possible consumption of explosives. This is quite a secondary consideration with the miner, though in its way worthy of most careful consideration. The miner's object is to raise the rock to the surface and extract its contents with the minimum total costs per ton, and explosives are only one item of costs. So we need beware when we see theories laid down solemnly, *ex cathedrâ*, and without modification, as, for instance, in regard to the right length of hole to be bored in certain work; for no attention is given to a number of vital considerations relative to saving time, and therefore to total cost of the work to be done, nor to several obvious methods of evading in practice the logical conclusions that can in theory be drawn from certain mathematically proved theorems. I do not think I shall be wasting the time of this Society if I bring forward some matters of interest met with in going over these works, and in also reviewing briefly the theory of blasting as therein laid down. It might then be possible to see if these theories can be applied with useful results under local conditions, and if they point out any directions in which economy can be gained in the use of explosives, I think I can make out a very good case for the more systematic study of this subject, both in our technical schools and by properly conducted experiments under actual mining conditions.

---

* *The Engineering and Mining Journal*, Sept. 19, 1908, p. 560.

† Transactions American Institute of Mining Engineers, February, 1899.

Another crying need of the industry is a printed sheet to be posted up on all mines, giving a simple and, as far as possible, non-technical résumé of everything we know regarding the employment of high explosives in boreholes to break rock, and pointing out the mistakes miners so often make, and the reason why they are mistakes. Such a sheet of instructions would, I believe, pay for its cost of preparation and printing in a month, in increased efficiency.

In a paper read before this Society nearly three years ago, I made some observations on blasting practice, and I tried to evolve some general rules drawn from my own practical experience. Apparently I did very little else but get myself into trouble as a dangerous heretic, and got myself pulverised by a special committee; but really, after all, nobody seemed to know much about the subject. I am ashamed to confess that at that time I had read nothing of the theory of blasting. I felt certain, however, that given the assumption of a homogeneous rock or one having well defined heads or partings with charges of known weight placed in chambers of known dimensions, the subject could be treated mathematically. I have since found that the few laws I guessed at regarding direction of holes and charges of explosives in relation to the burden, and of burden in relation to free face were in the main correct. Readers of books on blasting will find each writer saying the theory of the others is wrong. Gillette in his "Rock Excavation" makes some shrewd observations, but the "Blasting of Rock" by A. and Z. Daw,* if read in the light of practical experience and by people ready to break every commandment laid down in this blasters' decalogue, if they can thereby save time and money, is, I think, a valuable book. I will try and tear out the gist of it, leaving out the mathematical proofs and that part dealing with quarrying. Ten conditions are first laid down which influence the force and effect of a blast. They are, presuming that a hole has been bored, a charge inserted, the hole tamped and the charge detonated :—

(1) The size and number of the free faces presented by the rock mass. For instance, a drive has one free face only, a stope has two, a bench on an open cut, after the centre hole of a row has gone, has three.

(2) The tenacity or cohesive strength of the rock (available to resist rupture by shearing).

(3) The structure of the rock, whether jointed, massive, laminated, stratified or fissured.

(4) The strength and nature of the explosive compound.

(5) The character of the fuse and tamping.

(6) The thermal conductivity of the rock, and, I might add, of the tamping.

(7) Whether the blast acts alone, or simultaneously with others.

(8) Whether the rock falls when broken, or has to be lifted.

(9) The specific gravity of the rock.

(10) The size and form of the chamber.

(11) One might also add that a blast is influenced by the length of the line of resistance, in proportion to that of the height of the free face and of the length of the hole itself.

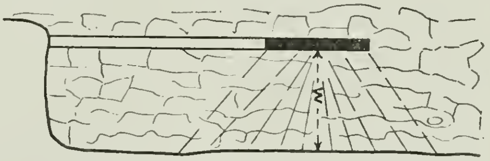


FIG. I.

Daw proves without difficulty that the old time honoured formula $L = CW^3$ where L represents the weight of charge (quantity of explosive) necessary, and where W = the line of resistance or the shortest distance from the charge to the nearest free face of rock, and C = a coefficient found from experiment representing the relative resistance of the particular rock to rupture, leaves out of consideration most of these factors and is useless when considered by itself. The force generated by the detonation of explosives, to be successful, must overcome

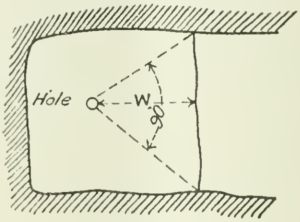
---

* The Blasting of Rock in Mines, Quarries, Tunnels, etc.: by A. W. Daw and Z. W. Daw. (E. and F. N. Spon, 153.)

(*a*) the resistance due to the cohesion of the rock tending to resist rupture; (*b*) the resistance due to the mass or weight of the rock. This is not relatively important in stoping, and where the rock is shot down it even assists rupture; (*c*) the resistance due to the jambing or hanging of the rock pieces together and along the lines of fracture. The force must act at 90° to the free face for maximum results. The force exerted by a blast on the rock must be a shearing force and not a bending or stretching one, because the explosive is in a small chamber and its force is suddenly applied to an inelastic rock mass. The force required to produce rupture by shearing, according to the theory of mechanics, where P = force required to produce rupture and S denotes the periphery of the chamber in which the explosive is placed, W equals the line of resistance and K_1 = a factor that represents the comparative resistance of that rock to shearing as determined, say, in laboratory in ft. lb. per sq. in. = the modulus of shearing for the particular rock. Then $P = SWK_1$.

Daw made experiments in ice, and proves that this formula holds good for gradual rupture, and he proves that suddenly applied forces produce similar results. The question that he seems to have neglected to investigate is, what effect varying the size and shape of the free face have in regard to the other factors. In his experiments the area of free face is varied within very narrow limits. He never defines a free face. This, as I will endeavour to show later, seems to me a serious omission, when we wish to apply this formula to actual mining. For instance, we have a hole of $1\frac{1}{4}$ in. diameter bored in the face of a stope, which is 6 ft. high. The hole is 6 ft. deep, bored parallel to the face of the bench. The burden on the hole, which is the line of resistance = W is 3 ft. The charge occupies $2\frac{1}{2}$ ft. of the hole. $P = SWK_1 = (1\frac{1}{4}\text{ in.} + 30\text{ in.} + 1\frac{1}{4}\text{ in.}) \times 36\text{ in.} \times$ modulus of rupture of quartzite to shear. The area of the free face at right angles to the line of resistance W is then $6 \times 6 = 36$ sq ft. Take the same hole and the same charge in a stope only 3 ft. high, according to the formula, the effect should be the same. We know very well, however, that the first hole would break and the second one would never break. The area of free face being 6×3, or 18 sq. ft. in the second case. According to the authors, the rock should apparently shear in a plane parallel to W as readily as it does to form the usual frustum of a pyramid with fracture planes at 45° to W.

With due deference to the authors, I would suggest that this formula is not true or satisfactory as thus stated. It is true only when the height and length of the free face bear a certain ratio to W and to S, so that the limiting lines of fracture set off from the perimeter of the sides of the chamber at an angle of 45° fall within the area of the free face. I will return to this later on. The authors then point out that with two free faces available (as in stoping) two portions of the rock may be ruptured off by shearing. "Owing to the inelastic nature of rock and the sudden force applied, equal tension is produced in the rock parallel to the line of resistance for any section that may be blasted," and that the resistance to rupture of the cross section parallel to the line of the hole may be equal to the resistance to shearing, and should be so to prevent "bull-ringed" holes. If F represents the area of such a cross section and K the modulus of rupture of rock $P = FK \therefore FK = SWK_1$.



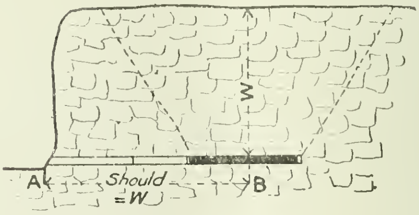


Fig. II.

Hence the authors argue that where there are two free faces, any hole should be given such a length in proportion to its burden, or W, that the rock lying between that portion sheared off directly in front of the charge, and the free face at the mouth of the hole will also be ruptured off. The force tending to produce rupture in blasting is proportional to the periphery of the chamber containing the explosive, such periphery taken at right angles to W, or line of resistance. "The section of rock that may be ruptured is proportional to the periphery of the chamber for a given line of resistance." It is owing to the condition that low explosives are employed to advantage in rocks of comparatively small cohesive strength, or where there are many lateral free faces and joints. These are used in large holes or bulled, or sprung, and are more economical than high explosives in small holes. Thus a mass of rock such as shown in Fig. III., with high explosives would merely break a crater, as the

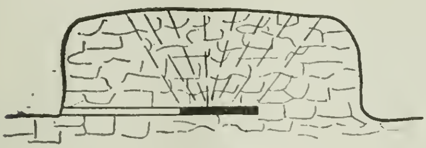


Fig. III.

cross section or the periphery at the top and bottom of the charge would be too small to give sufficient force to shear the rock along the hole and beyond it.

It seems to me that the authors have neglected the rupturing effect of gases from the charge escaping along the hole between the charge and the mouth of hole. This must, I think, have some effect in blasting; a larger effect with lower explosives, and a smaller effect with high explosives. Those who have seen a long hole fired have noted that the rock from the mouth is shot outwards quite as much as upwards, apparently by a force acting directly behind it. Regarding most mining here, we may, I think, neglect consideration of the resistance due to the weight of the rock blasted and the resistance or drag of the fractured rock to further movement, though occasionally in a stope we do see where a hole has had a charge large enough to fracture the rock without moving it. When previously endeavouring to estimate the increase of resistance to rupture of holes due to increase of burden, or increase of W, I hazarded the opinion based on personal experience that this increase varied as the square of the burden or W^2 for the same explosive. The authors prove this to be true, mathematically; the proof I will spare you. Now we see why as generally conducted breaking rock with hand holes in a stope, say, 48 in. wide, is much more economical in explosives than breaking the same ground with machines. The 6 ft. machine hole carries double the burden of the 3 ft. hand hole, but requires four times the explosive or more, and the question to consider is, cannot we bore and blast machine holes so as to reduce this difference? The authors say, "for the same explosive the resistance of cohesion to rupture in blasting varies as the square of the line of resistance." But here again we note that no consideration is apparently given to the size and shape of the free faces. Or what would happen if the face is only partially free, as in a narrow stope. The force developed by an explosive placed in a chamber in the rock depends on the following conditions :—

"(1) The absolute quantity of the gases produced. This in turn depends on the quantity and "power" of the explosive used, and I would add, the class or degree of detonation produced.

(2) The temperature of the gases at moment of detonation.

(3) The expansion of these gases due to heat liberated by the chemical combinations started by detonation.

(4) The time occupied in obtaining the maximum expansion or pressure.

(5) The size and form of the chamber.

(6) The thermal conductivity of the surrounding medium.

I would add to these—

(7) The amount and cohesion of tamping employed with certain explosives.

(8) The thermal conductivity of such tamping.

Explosives are of two classes—"low" or slow and rending, and "high" or quick and shattering. Of the first, gunpowder is the best known example. The explosives used here, gelignite, gelatine dynamite, and blasting gelatine belong to the latter class. In the former, chemical combination goes on comparatively slowly and gases are evolved gradually and at a low temperature. In the latter the substance is gasified almost instantaneously. "The full force of the gas is at once exerted in all directions and upon every part of the containing body; because motion requires time, and there is no time for the part that yields to move before full pressure is developed." This has given rise to the popular idea that dynamite, etc., acts downwards. Nitro-glycerine exerts a pressure on detonation of 12,000 kilo. per sq. centimetre, or about 25,000 lb. per sq. in., and blasting gelatine very little less. Only about 14% of the actual energy of the explosive is employed in doing useful work in shattering and displacing rock. This seems a very small proportion; but combustion or detonation is not always complete, gas escapes by holes and fissures, and the surrounding rock absorbs quite a lot of energy in the form of heat. Then the rock that is not displaced is "shocked," heated, and pulverised, and waves of force are sent through the surrounding rock and air. "If we assume that each unit of the same explosive compound will develop the same quantity of gases and attain the same maximum pressure under like conditions," two laws of the statics of fluids and gases help us to judge the relative force developed by an explosive. These laws are :—

(1) That the pressure exerted by a fluid upon the different parts of the walls of the containing chamber proportional to the areas of these parts.

(2) That the pressure exerted by a fluid in any direction upon a surface is proportional to the projection of the surface at 90° to the given direction.

Rock is inelastic, the limit being reached in most rocks with a slight change of form, therefore "there will be no appreciable enlargement of the chamber before rupture takes place." This is laid down by the authors ; but is this reasoning quite sound ? We know blasting enlarges the holes by reducing the walls surrounding the charge to an impalpable powder.

This is done either before rupture, during rupture, or after rupture. It cannot well be after rupture, and taking the case of a hole that fails to break and blows out one would think that the gases expend force over the enlarged area before they seek exit along the hole. What do others think ? The authors in speaking of cut holes seem to concede that such an action takes place. However, let us take this as proved. Then it follows that rupturing force P produced in any hole is equal to the maximum pressure produced multiplied by the projection of the chamber at right angles to direction of rupture = W. Therefore with the same explosive rupturing force is proportional to the cross sectional area of the chamber at 90° to W. And in comparing two holes having the same charges (which must entirely fill the chamber to develop their full power), that charge which is in a chamber having the greatest area at 90° to W in proportion to its size will do the most work. This is true, however, only within limits, as the thermal conductivity of the rock may absorb too much power in a very thin chamber. Hence, say the authors, $\frac{3}{4}$ in. should be the minimum thickness of a charge.

It will be seen that the cylindrical form of chamber as bored in rock by a drill is not the most economical. If we could put the charges we now use into chambers having a thickness parallel to the free face double that of their depth we could reduce explosives used by nearly 50%. This raises the question what is the effect of "chambering," "bulling," or springing holes in hard ground in our stopes ? It is here generally wasteful in explosives and not at all useful, as we require elongated charges for the best results. In very long holes in softer ground in quarrying and in very high stopes it is very useful. The authors then prove that in blasting tight rock where joints and fissues are not well developed, the diameters of boreholes should be directly proportional to the lines of resistance, and they state that they have proved this experimentally. Working with gelatine dynamite in strong granite which approximates the conditions here, they give the following table which is so interesting that I trust I am not acting unfairly in giving it nearly in full.

The diameters being here again proportional to the length of W.

On these fields this is a principle that is only partially observed in practice and which deserves further attention.

Diameter borehole.	Depth of borehole.		Length of Charge.	Weight of Charge.	Length of W line of resistance burden.	
in.	ft.	in.	in.	lb.	ft.	in.
$\frac{3}{4}$	3	2	9	·22	2	$4\frac{1}{2}$
$1\frac{1}{2}$	6	3	18	1·75	4	9
1	4	2	12	·50	3	2
2	8	4	24	4·20	5	4

In softer ground Eissler gave $1\frac{1}{4}$ in. dia. for 5 ft. (= W)
" " $1\frac{1}{2}$ in. dia. for 6 ft. (= W)
" " $1\frac{3}{4}$ in. dia. for 7 ft. (= W)

I pass over several interesting chapters on simultaneous blasts and turn to discuss what length, charges in cylindrical holes should bear to the diameter of hole where there are two free faces. The ratio is given as from 8 to 12 d where d = diameter of hole. If there are two free faces an elongated charge is the proper one to employ, not a concentrated one got by "bulling" or "chambering" the hole. "It is advantageous to have an elongated charge to insure the whole mass of rock being carried away to such free faces, as with a high value for $\frac{\lambda}{s}$ the blast would produce only a conical cavity." The authors also prove that, when we find, say, 8 in. of charge suits a hole 1 in. diameter to move a burden of 24 in., then if we increase the burden to 36 in. and use a $1\frac{1}{2}$ in. hole the same length of charge is the right one.

The authors next consider the best position for a chamber or charge when there are two free faces at right angles as in stoping. To obtain the best effect with a blast in rock there must be equilibrium of resistance on all sides of W to the action of the charge.

On this depends the right depth of hole to bore, say the authors, for any given burden. In hard ground, such as met with here, the distance from the centre of an elongated charge to the mouth of the hole at right angles to the line of resistance, as in a stope bench, should be equal to the burden or W., Fig. II. For example, we have a stope 6 ft. high ; the hole has a burden of 3 ft. : the charge will equal, say, 5 sticks of $1\frac{1}{4}$ in. gelatine ; the hole being $1\frac{1}{4}$ in. diameter at bottom ; the charge takes up, say, 30 in. of the hole. What, according to the author, is the right length of hole ? It would be 36 in. + 15 in. = 51 in. only. In practice we use holes 60 to 72 in. Hence, as I suggested, pressure of gas escaping along the hole must have some effect, or planes in the rock must greatly affect the result. Formulae are given for calculating borehole charges ; but as the coefficients C_a, etc., for the various rocks met with here are not available, it would be useless to check the calculations from actual practice.

The formula used for calculating charges where $L =$ weight of charge, is $L = C_v W^3$, and where the diameter of bore is known and charge used is *n* times the diameter, and the weight of 1 cub. in. of explosives in lb. = ·036 sp. gr. of explosive $L = \cdot 0283\ ngd^3$. The coefficient C_a in the formula $A = C_a SW$ and coefficient C_v in the formula $L = C_v W^3$ are found by trial shots. These are taken in average ground on average benches as nearly similar as possible. Say, 3 holes are bored with burdens of 2 ft., 2½ ft. and 3 ft., and are 1 in. diameter and are charged to a depth of 8 *d* with explosive. Then supposing the hole having a burden of 2½ ft. and a depth of 2 ft. 10 in. just breaks, throwing the broken rock only a short distance; from the equation $A = C_a SW$

$C_a = \frac{A}{SW}$. $A = 8$ square in. $S = 2\ (8 + 1)$ in.

Then $C_a = \frac{8}{18 \times 30} = \cdot 015$. $C = \cdot 02$ about for rocks such as are blasted here.

To find C_v in formula $L = C_v W^3$, $W = 2\frac{1}{2}$ ft. The weight of the charge L in ounces, taking the specific gravity of explosive (say, dynamite = 1·6), the weight of a cubic inch of dynamite = $\cdot 036 \times 1 \cdot 6 \times 16 = \cdot 9216$ oz. $\therefore$ $L = \cdot 7854 \times \cdot 9216 \times 8 \times (1 \text{ in.})^2 = 5 \cdot 79$ ozs. then as $W = 2\frac{1}{2}$ ft.

$$C_v = \frac{5 \cdot 79}{(2\frac{1}{2})^2} = \cdot 37$$

Theoretically these formulae should enable anyone working in homogeneous rock to calculate charges in all cases.

It must, however, as the authors point out, be remembered that benches vary in shape and that heads or planes of weakness are nearly always more or less developed. The art of blasting and rock drilling consists in noticing these and allowing sufficiently for their presence and in correctly forecasting what work each individual shot will do.

Regarding cut holes the authors in this paragraph appear to concede my contention that the enlargement of a hole during detonation of charge may affect the area over which the gases exert pressure. "The effect of breaking in a cut hole short is greatest if they meet so that the intervening rock is fissured and pulverised, as in that case a pressure surface for the gases from the explosion will be produced parallel to the face of the drive between the boreholes."

In a former paper I made some remarks regarding tamping. The authors quote the experiments of Gen. Sir J. F. Burgoyne with tamping powder holes. These seem to show that tamping owing to its inertia and friction on the sides offers far greater resistance to the escape of gases than I had thought. The General found that in 1 in. diam. holes 2 oz. or 4 oz. of powder would not blow out more than 7 in. of clay tamping, and when the blast breaks out its burden such tamping is often found intact in the hole. For high explosives I stated I did not consider the 12 or 18 in. of tamping often used by miners was necessary and considered a few inches to be sufficient. The authors say, in general it is sufficient to use a tamping of water, or a few inches of clay or paper pushed tightly home is all that is required. On the other hand, Mr. Rowland S. Oliver in the *Engineering and Mining Journal* of New York, writes:—"The fact is, however, that most blasters use an excess of explosive, so as to ensure breaking the ground, and this excess also makes up for loss of power by the escape of untamped gases." "Several years ago a manager in Arizona noticed that some miners complained of unbottomed holes and bad air. A 40% gelatine dynamite in ⅞ in. sticks and No. 5 detonators were used, powder being issued by the foreman. It was found one shift rammed charges and put tamping on top. The other shift did not tamp." The explosive used would, however, scarcely be called very high here, and the most important consideration seems to be to make the explosive completely fill the chamber to the exclusion of all air. "A ¼ oz. of No. II dynamite will throw a ball from a mortar 300 ft. If ½ in. air space is left between them it throws it only 210 ft., a loss of 30% of efficiency."

For this reason it is unwise to blow out holes too much before loading as a little mud among the cartridges expels any trapped air.

Efficiency and Tamping.—An opinion is gaining ground that water is not an efficient tamping for high explosives. Gillette quotes an authority as saying that it takes 3 lb. of dynamite with water tamping to do the work of 1 lb. with dry tamping. When we consider the enormous specific heat of water and its cooling effect on the gases produced this seems likely to be true. G. M. McFarlane, in the *Engineering and Mining Journal*, says, in discussing blasting long holes 15 to 20 ft. deep, "The remark is often made that water is the best tamping for dynamite. As a matter of fact, I note that in 'springing' or 'bulling' holes, water tamping is always blown out, even if the hole is full of water, whereas 7 or 8 ft. of sand tamping is seldom blown out, unless the rock is very tough and the bottom dead on the solid."

Development.—Here the authors have not much to say to help us. "In hard and tight rock the best length of advances (a length of round) with each set of holes is one-half the width of the heading." This means that we should bore holes only 3 to 4 ft. long in the faces of our drives.

"Experience shows that from ¾ to 1½ in. is the best diameter of hole to use in breaking hard ground," and this holds good here as elsewhere the diagram shown for the advances of a $6\frac{1}{2}$ x $6\frac{1}{2}$ heading shows 20 holes bored 3 ft. 4 in. long in the face of hard rock. These holes are $1\frac{1}{8}$ diameter and would be charged to a depth of 12 diameters with $\frac{3}{4}$ lb. gelatine dynamite, giving an expenditure of 15 lbs. explosive for an advance of about $2\frac{1}{2}$ to 3 ft. These 20 holes are, the authors tell us, fired in 5 rounds of 4 each. I pity the poor developer. I give this example to show how the mining student must beware of trusting to books instead of to actual experience. On the Rand, I suppose the most economical advance for such a heading would be 14 to 16 holes, 6 to 7 ft. long, $1\frac{1}{2}$ in. finishing, using about 50 lbs. blasting gelatine per round, and firing with two volleys only. The diagram for sinking a 14 x 7 shaft with 42 $3\frac{1}{2}$ ft. holes may be compared with figures given in Truscott's Rand Mining and practice in shafts here.

It must not, however, be imagined that these formulae and axioms are valueless. On the contrary, a knowledge of them rightly applied may save much time and money to a mine and its workers. And in tunnel and quarry work where a skilled engineer is available to direct the depth and direction of holes and their charge, large economies can be made. The idea, however, of a Rand miner sitting down to calculate his charges is, however, a vision of the far distant future. Nevertheless, it is true that the so called skilled workers of the Rand waste thousands of lbs. of explosives every month by boring and firing holes in direct opposition to all the correct principles of the art. Faces of drives have rounds hung up because the miner has neglected to bore enough holes to make W bear the proper proportion to the length of hole and the charge.

It is the relation of the theories mentioned to stoping holes that interests us most, and I hope we will have some help from our mathematically endowed members here.

There is one example of the application of these rules given in the book that bears on this matter. It is entitled "Economy of proportioning depth and diameter of borehole to height of bench of rock." Supposing it is found that a 1 in. hole $3\frac{1}{2}$ ft. long with burden or $W = 3\frac{1}{2}$ ft. with 2 free faces, *i.e.* (face of bench in which holes are bored and side of bench or stope face) when charged with 12 in. or ·543 lbs. of dynamite will blast a bench $3\frac{1}{2}$ ft. long (deep), is there economy if holes of 1 in. diameter are used for blasting a bench 6 ft. long. Holes required will be 6 ft. deep. W will be the same, and there must be the same length of tamping or vacant hole in each case, say the authors. Hence the charge in the long hole must be $3\frac{1}{2}$ feet long and must weigh 1·9 lbs.

Comparative volumes of rock broken will be $3{\cdot}5 \times 3^2 = 31$ cub. ft., and in second $6 \times 3^2 = 54$ cub. ft. Then in the both cases we have ·111 ft. of rock bored per cub. ft. blasted, but in the first case we use only ·017 lbs. explosive per cub. ft. blasted, and in the second ·0352 lb. or 104% more. "This example, say the authors, shows the importance of boring holes on correct principles to obtain greatest economy in explosives." In a recent number of our *Journal*, Mr. J. M. Phillips speaks of the trouble of bull ringing occurring in holes over 6 ft. long in stoping.

Now these examples are all right, and yet all wrong if applied to actual mining conditions. Here, with outputs to consider, economy of explosives is a secondary matter. Longer holes pay to bore in any case, as thereby less time is lost rigging up and adjusting larger machines.

Engineers have been asking if they would get any better drilling results in practice with higher air pressures. They would not unless pressures were increased enough to give an increased boring speed of 20% enabling 5 holes to be finished where 4 were drilled before, or unless miners stoping are provided with longer steel enabling them to lengthen their 4 holes as the pressure increased. It is indeed a striking commentary on the neglect that underground engineering on this field has suffered from in the past that we find leading engineers frankly confessing that they do not yet know what is the best pressure to employ in our mines to bore our rock.

Can we apply any of the foregoing formulae to stoping conditions here? Take this case. Height of stope 4 ft., no good heads or defined hanging or foot wall. What is the right diameter of hole to finish with? What is the maximum burden we dare place on such a hole in hard ground? What is the right length of hole to bore so that both faces are free faces, and so that the hole will break with a charge suitable for diameter of hole? If we add, so that the rock will be broken in the cheapest possible manner with due regard to all costs, costs of labour, power, explosives, capital costs and standing charges, we have here a problem as complex as any the human intellect has ever been asked to solve, one calling for the exercise of the highest faculties and involving an ever-widening circle of factors of various values, which must all be given their due weight. And it is solved largely by rule of thumb, or rather the solution is evaded. Yet some people call the mining problem here a simple one!

Now taking a roof hole H with $W = 4$ ft., it has the faces along HM and NH to break out, and theoretically for all the authors say should break direct along $HN = W = 4$ ft. by shearing as easily as along HN^1 in which case the stope would be 8 ft. high. We know that with a hole of

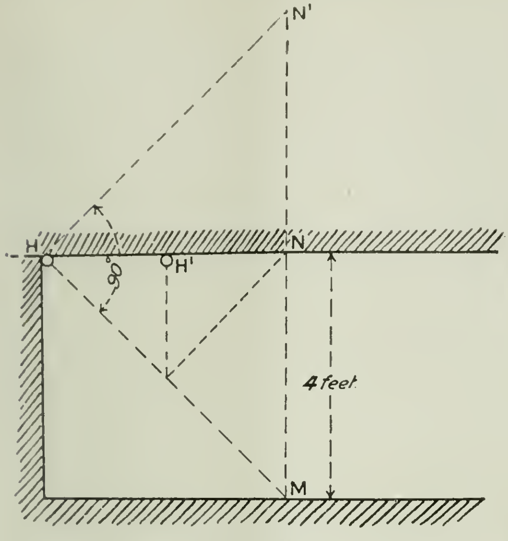


Fig. IV.

ordinary $1\frac{1}{4}$ or $1\frac{1}{2}$ in. diameter such a hole would never break though it would do so were the stope 8 feet high, Practice here seems to conflict with the authors' theory, unless as I suggest they have neglected to take into consideration the size and shape of "free faces."

In tight ground in such a stope about 24 in. would be the maximum burden, = W, given to such a first hole, and this corresponds to the greatest value of W, allowing the angle of the plane of cleavage to be 45° on each side of W. When there is a line of bedding, but not a properly defined hanging wall along W, then W might be made perhaps 3 ft., and were there a really well defined parting on the hanging W might even equal 4 ft. Theoretically, by using a large hole and a big charge any hole might be broken; but this is not true in practice.

Let us now consider the case of long holes versus short holes. According to the theory as shown, depths of holes must be directly proportioned to the resistance, burden or W, as the distance from centre of charge to mouth of hole should not exceed W. In practice here, this distance is generally $1\frac{1}{2}$ to $1\frac{3}{4}$ W., *i.e.*, a 6-ft. hole with a $2\frac{1}{2}$-ft. charge is given a burden = W = 3 ft. In narrow stopes, however, W is limited entirely by the height of the free face, as I have shown, unless the hole is bored a size that is uneconomical. Holes, any great depth, if fired in the usual way, do bull ring, *i.e.*, fail to break the rock near the collar or mouth of hole when a charge is placed in the end sufficient to overcome W. I was formerly severely brought to book for suggesting that where holes had to be bored by machines that take much time to set up and adjust, economy in drilling and in explosives would be gained by drilling long holes and distributing the charge into two portions by interposing tamping. It was contended that such a practice was dangerous, as leading to unexploded explosive being left in holes, and in any case in defective detonation.

I have since learnt to my chagrin that I was a martyr without even having the honour of being burned for starting a new and original heresy; for I have since found that Gillette, the classical authority on such matters, recommends the same thing for cutting out deep pipe trenches. I have no wish to open an old and "notorious" controversy; but by inserting a primer and detonator in each portion of the charge objections are overcome, and this way alone can long holes with small burdens be employed in narrow tight stopes. I wish to admit that if interposed tamping is used without any separate means of detonating the lower charge, this practice is open to the objections urged against it, and that detonation by concussion is not satisfactory. Lower grades of detonation are liable to result in waste of explosives and in the evolution of dangerous gases. It has been proved that even in a single charge 3 ft. long in a bore hole the bottom cartridges are sometimes not properly detonated owing to their distance from the detonator. If the smaller, one man drills, prove unsuitable for work here, and should a labour shortage again occur, recourse may then have to be had to $2\frac{1}{2}$ in. or $2\frac{3}{4}$ in. machines weighing 120—180 lb. for narrow stopes. These can be employed to the best advantage boring four to six long holes from one bench, holes being 5 to 7 ft. long, and finished off about $\frac{15}{16}$ in. in diameter. In these the charges would be distributed to gain economy in explosives. Another problem connected with the "easiest and simplest mining in the world" is, what is the right diameter to finish off hand holes, short machine holes in small stopes, and long machine holes in big stopes? Is the idea that explosive should "be where it belongs, at the bottom of the hole" an erroneous one in working most stopes, as the authors, I believe rightly, say it is?

Is the Committee of the Stope Drill Trials right in insisting that the limit of depth of holes should be 48 in., and that they should be $\frac{15}{16}$ in. diameter at the bottom?

Is one justified in stating that a 48-in. hand hammer hole, $\frac{7}{8}$ in. to 1 in. in diameter at bottom, breaks as much ground as two 36-in. holes?

Inter alia, might I suggest that it would be well worth while trying the experiment of increasing the weight of single-handed hammers given to boys putting down long holes. I think a 5 lb. hammer would give improved results in the hands of boys whose muscles were set; for it must be remembered that the percentage of the energy of the blow absorbed by the jumper increases greatly with long steel; might it not also pay in certain stopes to train boys to work double-handed, as in shaft sinking, and to fire a proportion of 60 in. holes? In stopes 48 in. wide and over, a very much heavier burden can be placed on a 48 in. hole than a 36 in. hole. If the burden on a 36 in. hole were 24 in., with a 48 in. hole and a 30 in. burden twice as much ground might be broken. Where the burden is limited by the height of the stope the gain is not so great. Hand holes 36 in. deep are now finished off about 1 in. in diameter. Might it not tend to economy in explosives to finish them with $\frac{3}{4}$ in. steel about $\frac{1}{4}$ in. smaller? If the force of the blast depends on the sectional area of the hole at right angles to the line of resistance, then a charge 12 in. long distributed along a hole $\frac{3}{4}$ in diameter would be as effective as one of 8 in. distributed along a 1 in. hole. It would contain a cubic in. less of gelatine, and if we are wasting this cub. in. per hole now, it amounts to a large sum per annum.

If the length of charge should bear a definite ratio to the length of hole then the length of charge can thus be best increased by decreasing the diameter of the hole within limits. The larger we bore our holes with machines or hand drilling, and the greater diameter we give them above a certain right amount sufficient to give gas enough to exert proper pressure on the walls of the chamber, the more time and money we waste in boring the holes and in the amount of explosive used. This amount increases with the square of the diameter without increasing its effectiveness at a like ratio, and the shorter the charges we get in our holes the more we must limit their length for fear that "bull-ringing" will be caused. I had intended to make out a number of boring, loading and blasting maxims and hints, but my paper is already too long, so I hope to avail myself of the results of the criticisms and suggestions I expect, and embody the list in my reply. I should like remarks on the question: In a four-hole bench with a well defined parting on (1) hanging-wall, (2) on foot-wall, which hole should be fired first? What should be the finishing diameter of holes bored in stopes 3, 4, 5, 6, 7 feet high (1) hard and tight and (2) fairly soft, breaking to heads, and on foot and on hanging wall to a parting.

The President: I wish to propose a very hearty vote of thanks to Mr. Weston for having given us this excellent paper; I hope that our mining members and those dealing with explosives will go into the matter thoroughly and give us the advantage of their knowledge in the discussion. There is very little doubt that economy may be effected in the use of explosives, and the more intelligently they are used the better it will be for the industry generally. It is not only a matter of economy, but the less explosives we can do with the less the pollution of the air in the mines.

Mr. W. Cullen (*Past-President*): The more mining costs are reduced the greater chance will there be for us manufacturers to get our explosives more generally used. I think the author deserves very great credit for the courage he has shown in tackling a subject which most mining men have consistently avoided. He says, in his opening remarks, "I think I can make out a very good case, etc." I agree with him heartily, and I think there is a very big field for anyone who cares to take up this matter. There is one point to which he refers, namely, that of inserting tamping between the different parts of a charge. I think the balance of proof is against him, but that proof, to my mind, has not been quite sufficient. Mr. Weston has mentioned a great many points which concern the practical miner and mining engineer. On these matters I do not propose to touch. Incidentally, my own paper which follows this will go over a portion of the ground he has touched on to-night; I hope to enlarge on it, but there is one subject, namely, that of tamping, on which there is a great difference of opinion even to this day. An abstract of a paper on this subject by a well-known mining engineer was recently submitted to the Publications Committee of the *Journal*, and, strange to say, this engineer deprecated the practice of tamping altogether. I am personally a very strong believer in tamping, and with the best we can get; I do not mean ordinary little cartridges of sand, which is but friable stuff and easily pulverised. My idea is that tamping should be more of the nature of a clayey substance. Actual experiments conducted within the past two or three months are against the theory, but the experiments have not got sufficiently far to demonstrate which practice is right, but it is my opinion, an opinion confirmed by a good many practical men, that improvements can be made on the system of tamping on these fields. I desire to second the vote of thanks to Mr. Weston for his very excellent paper.

THE ADAIR-USHER PROCESS.

(*Read at May Meeting, 1908.*)

By ALFRED ADAIR (Member.)

DISCUSSION.

Mr. A. Adair (*Member*): My former chaffing reply to Mr. Melvill, does not appear to have been understood by him seeing that he repeats his original statement at the last meeting, so I now give figures.

A sample of neutral slimes was mixed with 3½ times its own dry weight of water, cyanide and lime water. The solution contained ·025% KCy and ·022 alkali (as CaO): imitating works' conditions. The sample was thoroughly mixed, and evenly divided between two stoppered bottles of same size and shape. To one nothing was added, to the other ¼% of the weight of dry slimes of crushed umber, in its natural state. The bottles were shaken together several times during four hours, and After 16 hours they were allowed to settle, and after the night, shaken again several times. equal quantities of the clear solution decanted, filtered and tested for alkalinity, with $N/10$ acid and phenolphthalein with the usual precautions. Absolutely not the slightest difference could be detected between the two solutions, thus showing that both Mr. Melvill and Mr. Alexander must look elsewhere than to umber for their acidity.

Mr. Melvill now admits that umber is not acid, but still blames it for reducing his alkali. I believe neither of these gentlemen has any practical chemical knowledge, and therefore the statements about the acidity of a compound containing calcium carbonate are not to be wondered at.

Mr. Alexander said umber turned his solutions red and acid. He is apparently not aware of the true and quite evident explanation, that umber had nothing whatever to do with it, but that somehow or somewhere, he had allowed sulphuric acid to get into his solution ; the red colour was due to HSCN and a slight amount of ferric sulphate.

Mr. E. H. Croghan (*Member of Council*) : I think that the discussion which has taken place up to the present proves the value of the very interesting paper submitted by Mr. Adair on the Adair-Usher process. The reply to the criticisms will be still more interesting. The important bearing this process has had and still has on the gold industry needs no comment. I have no doubt that there are still many improvements to be made, but the trend seems principally in the direction of mechanical appliances. Chemical factors are practically at a standstill until perhaps one day certain chemicals, which are known to be of great assistance, can be manufactured at such a cost as it will ensure their cheap commercial applications.

It had been my intention previously to say a few words on the peculiar properties exhibited by umber in our industry, but Mr. Melvill's remarks on his practical experience with umber partially stopped me. In view of the author's contributed criticism on the subject of Mr. Melvill's query, I feel I should bring forward some experimental data I personally obtained with umber. Some time back, trials were necessitated in consequence of its use on a large scale affecting cyanide solutions. The only observation noted at the time was the very appreciable decrease in alkalinity that took place. I therefore made some laboratory experiments on the alkalinity supposition, and to my surprise found that consumption of alkali was very apparent. The experiments were carried out in closed vessels and under such conditions as precluded the possibility of any interfering factors. As I have not been able to find the figure records, the preceding remarks would be of little value without confirmation. In consequence I recently made another series, but proceeded a few steps further, as time permitted, to note whether umber affected the free cyanide, the total alkalinity and the total cyanide. The cyanide solution employed was that leaving the strong boxes.

The details of the experiments are as follows:—A definite volume (500 c.c.) of the cyanide solution was poured in each case into ordinary colourless wide-mouthed glass-stoppered bottles. To the first nothing was added, it being the blank or original solution. To the others so much raw umber was added as to constitute additions equivalent to 2%, 1% and 0·4%. They were then stoppered, shaken a few times for a minute or so, and allowed to remain in contact for about 18 hours, when they were each once more given a shake. The solutions were then filtered off on dry filters, and the estimations at once proceeded with. During the whole period of 18 hours the stoppers were never removed nor were the solutions unduly exposed to light.

The percentage data obtained are as follows, and are expressed in terms of the potassium base.

	Blank.	2% Umber.	1% Umber.	·4% Umber.
Free cyanide ...	0·088	0·079	0·079	0·088
Protective alkali	0·067	0·002	0·013	0·040
Total cyanide ...	0·156	0·145	0·151	0·151
Total alkali ...	0·215	0·130	0·152	0·197

The original cyanide solution contained in a non-actinic bottle, and the blank as before stated were in perfect agreement after 18 hours, and no differences were noted with those data when the original solution was first tested a few hours after its arrival. The umber treated solutions acquired a distinct greenish tint as compared with the original colourless solution. Curiously some hours afterwards this tint disappeared, and indicates oxidation of some description as the balance of the solutions not required were simply left in open Erlenmeyer flasks.

The preceding results are in my opinion extremely interesting, and more so from my point of view as they are an original publication. The data obtained for the free cyanide indicate that a slight decrease takes place if strongly umbered, but is not affected with smaller quantities, such as 0·4%. The other results are very definite, and I am firmly convinced that not only is the protective alkali affected, but also the total cyanide and alkali. The explanations that might be submitted are of no avail without detailed examinations of the liquors. It is my regret that I cannot undertake the research, for I feel assured some very interesting data would result not only on the action of umber on cyanide solutions, but possibly also on amalgamation. It would be still more valuable if the investigations were carried out on fresh umber, and after it had been exposed to atmospheric conditions for a week or so.

In conclusion, I disagree with the author that to a trained chemist the analysis of umber should be a sufficient reply, because in my opinion local umber has not been examined in such a manner as to correctly define the exact state in which the constituents composing it exist. It is one matter to hold practice up to ridicule but it is a horse of another colour to explain it theoretically. It is a curious thing that many of the most important discoveries in science have been made not by men of science, but by ordinary practical workers who are generally not endowed with the so-called genius of theory. The umber case reveals the old adage that scientific truths are often brought home in the first instance by practical results, and in this case the action of umber can also be proved by laboratory experiment.

A LABORATORY COMPARISON OF TUBE MILL PEBBLES.

(Read at June Meeting, 1908.)

By Prof. G. H. Stanley, A.R.S.M., M.I.M.E., M.I.M.M. (Member of Council).

With an Appendix on Liners, by M. Weber.

DISCUSSION.

The President: We must regret that up to the present there has been no discussion upon this paper. But perhaps to discuss the results of Prof. Stanley's investigations would require more petrological knowledge than most of us possess and, further, the necessary time and means for cutting sections of material and the microscopical apparatus for examining them is wanting. Given these, the information which Prof. Stanley and Mr. Weber have placed before us should prove a useful guide in selecting a suitable material for liners. The use of pebbles here has now been almost entirely superseded by banket, so that though the author's results, from a scientific point of view, are very interesting, to us on the Rand their practical value in this respect is small. To those, however, who are tube milling in places where the ore itself is not suitable as a grinding agent, the information given in the paper should prove of the greatest value in determining the choice of a suitable agent for grinding purposes. The question of the best material for liners will always interest us, and I trust that the author will always let the members of our Society have the benefit of any further investigations he may make on this subject, which is of great interest to all those who have charge of tube mills. I have to thank Prof. Stanley and Mr. Weber for their contributions.

BATTERY AND CYANIDE GOLD SMELTING.

(Read at July Meeting, 1908.)

By A. Thomas (Member).

DISCUSSION.

Mr. L. J. Wilmoth (*Member*): Towards the end of his paper the author gives some assay results of his residual slag from the pan furnace. It may be taken that the assays of these residual slags are an indication of the work done, and therefore can be used as a standard of comparison. The value, however, of any assay result depends entirely upon its accuracy, and with your permission I would like to draw the attention of assayers to some of the difficulties in obtaining a true valuation of this residual slag.

These few remarks, and the results of a few experiments, I would like to give, only apply to the residual slags obtained from the smelting of such materials as the author describes in his paper. It would seem from the work I have done that the crucible assay is the most accurate method of dealing with them. In these slags the silver values are high, and therefore its accurate determination is necessary. This determination, unfortunately, is complicated by the fact that the fluxes which yield the highest results for the gold give consistently low results for the silver.

My first series of experiments were carried out on a slag of very high gold values, in fact, so high that it paid for re-treatment.

Experiment I.—After fine grinding and very thorough mixing I took six portions of ·2 A.T. and scorified them in 3-in. scorifiers with granulated lead and a pinch of borax. The average result of these six assays were :—

Gold, 31·8 dwt. ; silver, 182·5 dwt.

Experiment II.—I took four portions of the same sample and fused it in a G crucible with the following charge, which was poured without any wash of litharge and charcoal :—

Slag, 1·0 A.T. ; litharge, 1·0 A.T. ; borax, 1·0 A.T. ; soda, 0·5 A.T. ; charcoal, 1·7 gm.

The charcoal used is high, but the button obtained was only normal. This is accounted for by the highly oxidising nature of the slag, due no doubt, to the higher oxides of iron present. The slag was perfectly fluid on pouring, but during fusion showed a great tendency to boil or froth, thus necessitating the use of so large a crucible for so small a charge. The result of these assays were :—

Gold, 30·2 dwt. ; silver, 184·4 dwt.

Experiment III.—I then took another batch of four assays and used a charge carrying 2 A.T. of litharge instead of the 1 A.T. as in the last batch, the charge being :—

Slag, 1·0 A.T. ; litharge, 2·0 A.T. ; borax, 1·0 A.T. ; soda, 0·5 A.T. ; charcoal, 1·7 gm.

These charges were also poured without any wash of litharge and charcoal. The boiling was excessive but does not account for the following low results :—

Gold, 28·0 dwt. ; silver, 121·8 dwt.

Experiment IV.—I now turned my attention to the effect of a wash with litharge and charcoal just before pouring, using the same charge as described in Experiment II., the results being as follows :—Gold, 32·4 dwt. ; silver, 199·0 dwt.

Experiment V.—Using the same charge as in Experiment III., only giving a wash of litharge and charcoal before pouring, I obtained the following average results :—

Gold, 33·8 dwt. ; silver, 196·6 dwt.

The following table gives the results of the experiments with the varying proportions of litharge, with the wash and without it :—

COMPARISON OF RESULTS.

Assay.	Gold.	Silver.	Proportion of Gold to Silver.
	Dwt.	Dwt.	
Scorification ...	31·8	182·5	1 : 5·74
Crucible using 1 A.T. litharge, no wash ...	30·2	184·4	1 : 6·10
Crucible using 2 A.T. litharge, no wash ...	28·0	121·8	1 : 4·35
Crucible using 1 A.T. litharge, with wash	32·4	199·0	1 : 6·14
Crucible using 2 A.T. litharge, with wash	33·8	196·6	1 : 5·81

These comparisons are interesting, as they show the effect of the use of litharge on the assay values. Of course, it is not new, but I would like to point out and demonstrate the fact, as on the Rand, with our low silver value in the ore, we use a large proportion of litharge in our routine assaying, and so may be tempted to do so on these odd samples.

Looking closely at the table we see that the charge using 2 A.T. litharge and receiving a wash gives the highest values for the gold, whilst the 1 A.T. litharge charge with a wash is best for the silver, plainly showing that, though a wash does clean the slag, in this case the silver, owing to its state of oxidation, cannot be completely recovered. A comparison of the results and the proportions of gold to silver in the charges using 1·0 A.T. litharge show that, though the washed assay gives the higher results for both values, it does not alter the proportion of the two metals, thereby showing the absence of the oxidising effect of the low litharge on the silver.

With your permission I would like to go on now to the consideration of the effect of borax on this assay. The subject is highly interesting to us assayers, in view of the wordy warfare that has waged in this Society on this subject for some time past. Here again, I would mention that these remarks apply only to the residual slags obtained from the smelting of such material as the author mentions, and so must not be taken as an attempt to stir up further strife on that much vexed question, the use of borax in the assay of banket ore. These experiments were carried out some months later than those illustrating the effect of litharge, and so the sample is different. The values were high in silver and moderately so in gold. The sample after fine grinding was roughly mixed and then divided into two portions. As will be seen, the different portions differ slightly in values, but this does not affect the conclusions arrived at.

Experiment VI.—One portion of the sample was most thoroughly mixed, and eight portions taken for assay. Four were fluxed, as in charge A, and four as in charge B. The following are the two charges :—

A.—Slag, 1·0 A.T.; litharge, 1·0 A.T.; borax, 1·0 A.T.; soda, 0·5 A.T.; Charcoal, 1·7 gm.

B.—Slag, 1·0 A.T.; litharge, 1·0 A.T.; soda, 1·5 A.T.; charcoal, 1·7 gm.

These were run down without any wash before pouring. Charge A is exactly the same as used in Experiment II. Of the charges, A fused down very quickly and B, on account of the frothing of the soda, was some time in coming to the state of quick fusion. Charges A all poured with great fluidity whilst charges B were all pasty. The lead buttons from charges A—as is usual with a new borax flux—broke away cleanly from the slag. The following are the results of these assays:—

Charges A.—Gold, 9·6 dwt.; silver, 195·6 dwt.

Charges B.—Gold, 8·2 dwt.; silver, 201·4 dwt. It will be seen, the non-borax flux gave a lower result for the gold and an appreciably higher result for the silver. As a check on these results the other portion of the sample was taken and the experiment repeated. The results were as follows:

Charges A—Gold, 11·66 dwt.; silver, 200·8 dwt.

Charges B—Gold, 11·26 dwt.; silver, 208·0 dwt. There would, therefore, seem to be very little doubt as to the effect of the borax on the silver and the effect of its absence on the gold values. Thinking a wash before pouring might overcome this, I put through another batch of eight samples—four fluxed as in A and four as in B—and the results certainly did not improve in regard to the silver, though the gold values went up in each case, as the following results show:—

Charges A, with a wash.—Gold, 12·60 dwt.; silver, 201·0 dwt.

Charges B, with a wash.—Gold, 12·00 dwt.; silver, 205·2 dwt.

The low results for the gold in charges B, I attribute to the lack of fluidity due to the absence of borax. A glance over these results would seem to show that we cannot obtain a really accurate determination of the two metals in the one fusion. For the estimation of the gold contents a charge using high litharge and borax, as in Experiment III., and using a wash before pouring, would seem to be the best, whilst a separate assay of the sample should be made for the determination of the silver. In this charge only enough litharge to give the desired button, and no borax should be used.

I must apologise for having drifted off into a side issue of the author's paper, but, considering the necessity for the accurate valuation of these slags before comparisons can be made, I feel myself more or less justified in raising these few points.

Mr. A. McA. Johnston (*Vice-President*): The author mentions in his paper that he has every reason to believe that gold is carried off to the flue from the reverberatory chamber in which the gold-zinc slimes are smelted. His conclusion is based, I believe, on the fact that the slag formed on the inside brick lining of the flue has, in time, become dotted with small shots of gold. I would, however, join issue with him in his remark, chiefly because of its vagueness—a vagueness which Mr. H. A. White tries to obliterate by mentioning a 3% risk of loss.

Some few months ago, some doubt existed in the mind of the consulting metallurgist of the Consolidated Goldfields *re* the loss of gold in the particular type of furnace described by the author, where no condensing chamber existed. I carried out tests to discover, if possible, the actual loss of gold and had to come to the conclusion that approximately during the smelting of thirty-seven No. 100 pots the loss mechanically and by volatilisation amounted to 7s. That this small loss is due to the method adopted by the author of placing his flux on the top of the calcined slimes, I have little doubt, and this conclusion is partly borne out by the fact that the gold and the silver obtained in the experiment gave an alloy containing about 25% gold and 75% silver, a proportion not found in cyanide gold on these fields. The loss therefore must have been due mainly to volatilisation during smelting with practically no mechanical loss. A small piece of porcelain tube which had fallen into the flue and had remained there about three hours, the time taken by a "smelt," was found to be covered with slag. This was wholly crushed and gave an assay value of 42 dwt. of fine gold and 24 dwt. fine silver per ton. These results bear out, roughly, the figures obtained at the Ferreira in the bag house experiments detailed in these pages by Mr. Rusden.

Mr. H. A. White is inclined to belittle the dry calcining and pot smelting method by shielding himself behind the remark of a Past-President. Without going into the merits of this statement as being based on less scientific methods than now in use, I think Mr. White rather neglects from his calculation one step in the treatment of these zinc slimes. Most of us know that the process consisted years ago of drying and calcining, followed by the grinding of this product, then mixing this ground product with the flux, charging the mixture into the pots and afterwards smelting in the reverberatory. Figures as to probable loss of gold in all of these operations I have not, but anyone acquainted with the actual method of procedure on these lines will, I am sure, agree with me that the grinding and subsequent mixing of the calcined slimes with the fluxes is the operation where loss would be heaviest. That a loss of 3% however is brought about, I am loth to believe, unless unfair conditions or carelessness exist, and I feel sorry that Mr. White should have resuscitated this statement without adducing actual figures whereby more authority could have been given to it. Observant members will have noted that these operations necessitating dusting have been eliminated by the author and so far as actual results go, I see no reason why other smelters should not follow on these lines and bury our 3% "risk of" loss.

The point raised by the author regarding the depth of purple colour varying with the richness of the slime is interesting, and I should like to hear from some of our other smelters if this is so. Personally, I am rather inclined to think that the depth of colour varies with the richness of the slimes, and that the white lime used only serves to throw into relief the varying gradations of colour. A well treated and thoroughly washed slime will give a richer purple colour than one which has been badly acid treated and washed.

With Mr. White I am in agreement *re* the size of trays to be used in the retort. Occasionally I have seen a little injudicious stoking lead to over heating of the retort, and the consequent slogging and chipping required to get the melted gold out of these angles. I remember once assisting in getting out about 80 oz. of fine gold from some old retort trays which had been stored away by the previous smelter. The most of it was smelted into these sharp corners and after chiselling out as much as we could, even at the expense of the trays, we separated the remainder by a prolonged contact in a solution of sal ammoniac with, as I said before, a recovery well worth the little trouble and expense involved, and some additional "kudos" to the battery manager.

Prof. G. H. Stanley (*Member of Council*): With regard to the colour of these calcined slimes I once made some experiments which necessitated calcination with a poor sample containing only 2½% gold, and the colour resulting was a dark red.

The President: I can corroborate that; I have seen exactly the same thing.

REMINISCENCES OF THE EARLY RAND.

(*Read at August Meeting, 1908*).

By M. H. Coombe (Member).

DISCUSSION.

Mr. J. S. Curtis (*Member*): I have read the author's paper with great interest. He has given a most graphic and entertaining account of the conditions prevailing on the Rand in the early part of 1887, and, with a minor exception or two, I can fully corroborate what he says.

I came from Barberton to the Rand in the latter part of March of that year, and must have arrived here about the same time that the author did from Kimberley. Mr. Gardner Williams and I had trekked through from Barberton to Pretoria in our ox waggon, and after a short stay in the capital, giving us time to hire a fresh span of oxen, came on to the Rand and made our first outspan at Du Preez. The next move was to "Knights," where a hundred stamp mill was in course of erection. From there we continued our journey along the Main Reef, examining the various developments, and on the second day about noon arrived in the Market Square. As the President has pointed out, there was a two-storey building going up on the North side of the square, and this must have been Henwoods, but as it was not yet in full swing we repaired to Booth & Co.'s, at the south-eastern corner of the square, to replenish our provisions. At that time there might have been a few other small buildings on the square, but there were only a few, at any rate. On Commissioner Street the Central Hotel was doing business, and the foundation of Height's was being laid. There was Lawrence, the chemist, dispensing medicines from a tent, and there were a few iron buildings, mostly occupied by attorneys. All these were on the south side of the street. On the north side of the street, if my memory does not fail me, there was only Donovan Bros.'s stables.

In May of the same year I was engaged as engineer by Mr. J. B. Robinson (now Sir J. B. Robinson) to open up the ground of the Robinson Syndicate, which comprised, among other valuable properties, the present Langlaagte Estate and the Robinson Mine. I may add that in opening up this ground I also opened my eyes as well to the prospective value of the Main Reef.

As the author has mentioned the principal workings along the reef, I do not think it necessary to speak of them further.

The only points upon which the author and I do not agree are the rates of wages for white miners and natives prevailing at that time. At the Robinson, Crown Reef and Langlaagte the wages then were:—For white miners, £5 per week, and for natives, 7s. 6d. per week. The native wages, owing to the scarcity of boys, soon reached 10s. per week and, if I am not mistaken, white miners' wages advanced to £1 per shift before the end of the year. In talking the matter over the other day with Mr. Godfrey Lys, who was then managing director of the Crown Reef, he fully agreed with me as to the rate of wages paid. That there were other rates of wages prevailing in other parts of the Rand at the time the author mentions is quite possible, but they were not those paid at the mines of which I have spoken.

Mr. W. A. Caldecott (*Past President*): Whilst my personal remembrances of the Rand do not extend quite so far back as those of Mr. H. M. Coombe, possibly one or two incidents of the early days may be of interest to members.

Having arrived at this camp by coach from Kimberley in 1889, I entered the mill of the Du Preez Company (now Rietfontein), as an unpaid learner of the art and mystery of the amalgamation process. That veteran member of our Society, Mr. J. S. Curtis, was general manager, and the battery, which had lately been completed, was considered a good-sized and up-to-date mill.

It contained 20 stamps, weighing 750 lb. each. Part of the ore came from an open cut, and was conveyed to the mill in Scotch carts. The walls of an old kraal on the outcrop of the reef were also milled, although ore carrying less than an ounce per ton was then considered rather low grade. No assay office existed on the company's property, the pan in the mine captain's or millman's hands being then customarily considered all that was necessary. Ore becoming scarce, the mill was shut down from 10 p.m. to 6 a.m., or whenever the slimy water overflowing the tailings dam was too small in quantity or too thick with slime to use. Our recovery was what we got off the plates as amalgam and deposited in the bank as retorted gold, and few troubled further. The present-day millman with his systematic sampling and assay returns, micrometer gauges for measuring screening wires, grading analyses and ratios of liquid to solid in pulp, alkali tests for mill service water, thermometers for pulp temperatures, and elaborate records culminating in a theoretical extraction and actual recovery, was indeed a product of the future. Passing by various personal experiences, I may confirm the author's statement as to the low rate of pay at that time for both natives and white men. The average monthly rate of pay of the amalgamators in our mill was about £20, and the assistant manager drew £35 a month, whilst living, without railways and only ox wagon transport of all goods, was by no means low. At that time the total output of the Rand was little more than the present output of the Simmer and Jack, and technical literature upon gold extraction work, which is now so abundant, hardly existed.

In the middle of 1890 I entered the service of the Cassel Gold Extraction Co., then beginning to develop the MacArthur-Forrest cyanide process, under the direction of the inventor, Mr. J. S. MacArthur, with whom it was my privilege in this way to become acquainted. Their demonstration plant, consisting of 1¼ tons agitation vats discharging into leachers of a little larger size was erected at the old Salisbury mill. The existence of gold in the reefs beyond a very moderate depth was then very gravely doubted, as also the possibility of recovering the gold, should the values persist into the "blue." "Pyritic" and "refractory" were then considered synonymous terms, notwithstanding that our amalgamation recovery at the present day from deep-level unweathered rock is far higher than from the oxidised and so-called free-milling rock of eighteen years ago. The claims of the cyanide process on its introduction here were viewed with the greatest incredulity, and were classed with those of many competing methods, which are now merely of historical interest. An experiment which I carried out while still in the Du Preez mill, on first hearing of the process, inclined me to share these doubts. It consisted in placing some visible gold panned off from the ore, together with water and a lump of mill cyanide, in a bottle which was lashed to a stamp. Next morning gold was still visible in the bottle, and the claims for the efficiency of the process were in my opinion seriously discounted!

Solutions were employed in the demonstration plant of 1% cyanide strength and upwards, but the remarkable differences in assay values before and after treatment of various parcels of material, combined with the weighty evidence of small bars of gold bullion actually produced in this demonstration plant, gradually aroused conviction. Possibly the greatest triumph was a very high recovery from a parcel of pyritic concentrates from the Percy mill, assaying some 25 oz. per ton. At a time when mining companies actually paid to have their tailings carted away lest their mills should be buried, the prospect that these could be treated at a greater profit than by buddle or vanner concentration, followed by chlorination, opened new vistas of hope. The records of those times when assays were made on ounces of material and weighed up in thousandths of a grain and reported in troy ounces, dwts. and grains per 2240 lb. ton, when cyanide strength determinations were carried out on 130 fluid grains of solution, and the volume of solutions applied to charges measured in gallons, would indeed appear distant, were it not that some of the former practices still linger in England, and our friends, the engineers, still prefer to speak of millions of gallons of water instead of employing the common cyaniding unit of a fluid ton of 32 cub. ft.

The demonstration at the Salisbury plant being over, I travelled by coach for three and a half days to the De Kaap district to cyanide the Sheba tailings in Fever Creek for the African Recovery Syndicate. This tailings dump contained 5,400 tons of 50 dwt. sand, every truck-load of which was weighed and sampled. The plant, which started work on the 14th February, 1891, comprised six wooden square leaching vats, each of 20 tons capacity, under a shed roof. As about 1,000 tons were worked monthly, the treatment circuit was about 3·6 days, 1% cyanide solution being used. Our working costs were some 30s. per ton, two-thirds of this being accounted for by a cyanide consumption per ton of 10 lb. of 70% cyanide at 2s. per lb. However, the recovery averaged 70%, so that the monthly profit was over £5,000, and everyone was very well satisfied. Needless to say, these residues have often been re-worked since! On these results and those at the Robinson G. M. Co.'s cyanide plant, of which Mr. G. A. Darling was manager, and which had started work a little earlier, the African Gold Recovery Co. was floated. A new era in the metallurgy of gold had been ushered in by this proof under regular working conditions of the economic value of the cyanide process.

I will not detain you by relating at length further experiences of these past times,* such as the proposals of the Barberton agent of the A.G.R. Syndicate to reduce capital expenditure by running the plant without the aid of an assay office, and to sluice the residues over amalgamated plates; ignorance of the fluxes required for smelting, with charcoal or coke at £30 per ton, the zinc-gold slimes; the use of tea-chest lead for assay cupellation and of improvised measures, others being unobtainable, in the shape of cut-down bottles for solution tests; failure with attempts to refine the zinc-box precipitate by acid treatment, and better success with a 10% cyanide solution for the same purpose; the observation of leaching pipes gold-coated inside through the strong rich solutions employed; attempts, which inexplicably failed, to accelerate the dissolving of the gold by mixing ground charcoal with the sand, so as to form a gold-carbon couple, and to replace cyanide as a solvent for gold by using cyanogen gas in conjunction with sodium hyposulphite solution; the discovery of some 300 oz. of gold in the deposit in the sumps, into which it had been mechanically carried through the rush of solution and violent chemical action in the little zinc boxes; unfortunate experiences with granulated zinc for precipitation, in place of the shavings turned from discs cut by hand from the zinc linings of the cyanide cases; the casting of thick zinc discs from scrap zinc and turning into shavings after annealing, when our supplies were cut off by flooded rivers; disappointment on being informed by Mr. MacArthur what I imagined to be a discovery in using lime for reducing cyanide consumption had already been patented by himself a year or two before; and a condemnation by the most prominent of the local metallurgical experts of the cyanide process because there was "no strength in it," as compared with the suffocating odour associated with vats containing chlorine. At that time the important part played by oxygen in the dissolving of the gold was by no means realised, and the solution storages in some of the early plants were covered to prevent an anticipated loss from cyanide being converted into cyanate through atmospheric oxidation. In truth, the cyanide process has succeeded not because of, but in spite of, the knowledge of some of its early exponents.

In reflecting upon the changes which have taken place in consequence of the introduction of the cyanide process eighteen years ago, the actual amount of gold, enormous though it is, annually won by this means only indicates part of the benefits derived. Nowadays, for the large majority of mines here the gold won by cyaniding represents the difference between profit and loss, and hence but for this secondary treatment only a tithe of the ore now milled could be profitably handled. To employees and the local business community the benefits derived from mining low-grade ore are as great as for rich ore, and thus the existence of the bulk of the population on these fields depends upon the results of cyaniding, which alone permits the majority of our mining companies to continue in operation. The prediction has indeed been amply fulfilled, "that cyanide of potassium, hitherto used only to polish amalgamated plates, will take a first rank as chief agent in gold extraction."*

Another result of the introduction of the cyanide process has been the development of scientific methods in gold ore treatment generally. The gold contents of material cyanided being usually invisible in the pan before treatment as well as after, necessitated constant sampling and assaying, and consequently a certain amount of gold was called for by these returns, instead of a mere acquiescence in the fact that at the end of the month a certain amount of gold was available for lodgment in the bank. An understanding of the reactions taking place in hydro-metallurgical operations required some knowledge of chemical principles, thus creating a demand for men with attainments beyond those of their hardy predecessors, the muscular metallurgists operating in Dead Dog Gulch or Wallaby Creek in past days. These tendencies have acted and re-acted in all branches of gold extraction work and to their effect, greatly aided by the free interchange of ideas in our own Society, may be very largely attributed the reduction of ore treatment costs and the high percentage recovery of values, which are such marked features of the last decade.

The President: In thinking over old times in Johannesburg the other day, I jotted down a few points. With regard to the Aurora property referred to by Mr. Coombe, if my memory serves me right, its first mill was built right over the hills to the south at Diepkloof. Later, a mill was erected on the range opposite Maraisburg. Many of the buildings then in Johannesburg were built of canvas with wooden frames, so that if the reef petered out they could just be packed on an ox wagon and shifted somwhere else, and it was certainly very amusing going through the old place at night when the inhabitants were, some of them, retiring to rest; there were some very curious shadow pictures. Then there was the Ferreira mill at San Souci. I very well remember seeing the ore carted from the open Ferreira workings in Scotch carts, and also water being got out of the cuttings with a Californian pump, consisting of a chain, with wooden discs at intervals, passing up a wooden pipe. As these discs came along they imprisoned a certain amount of water and carried it to the top. I recollect watching this stream of water running down the road and distinctly seeing little grains of gold trickling down with the stream, which showed how rich the outcrop of the reef was. Old sampling methods occur to me. I recall an instance where the mill manager was asked how it was that seeing that the rock supplied to his mill was, according to mine sample, of an average grade of 5 oz. to the ton, his recovery only amounted to 1 oz. It was pointed out, of course, that there was a certain amount recovered by cyanide (it was just in the early days of that process), and there was also a certain amount in the residues. There was no suggestion of thieving. Everything was granted to be perfectly correct, but it could not be understood how it was that the total recovery, plus residues, amounted to only a little over 1 oz. The question of sampling was then considered, and it was found that the sampling was being done by the mine captain who was what might be called a specimen hunter. It turned out that reef not showing visible gold was not sampled, so no wonder the value of the rock going to the mill assayed 5 oz., and the recovery was only 1 oz. Another method of sampling was to take a rock drill and bore a hole into the body of the reef and take what the drill brought out as a sample of the reef, making no allowance for stoping width or waste. That was supposed to be sampling and valuing the reef. When working at one time in a small mill in Klerksdorp it was very interesting at night to see the fire-flies, of which there was an enormous number. They came into the mill through the tail race until the whole place was illuminated with them. It was an exceedingly pretty sight.

---

* See this *Journal*, vol. i., August 21, 1894, p. 22; also the introduction to "Cyanide Practice," by Alfred James.

* J. S. MacArthur, *Journal of the Society of Chemical Industry*, March 31, 1890; see also this *Journal*, June, 1905, vol. v., p. 347.

SMALL MINES OF RHODESIA.

(Read at September Meeting, 1908.)

By B. I. Collings (*Member*).

DISCUSSION.

Mr. J. M. Phillips (*Member*): The author is the first person I have heard of who has cared to give even an approximate date of the old workings of Rhodesia.

I would like to say that in some of the old workings on the Rezende Co., Umtali, several stone implements were found; evidently hammer-heads and picks; these in conjunction with numerous signs of fire in the same place, which agent was, no doubt, employed to extract the reef, point, in my opinion, to a far longer period than 200 years having elapsed since the occupation of the land by the ancients. On a hill in the Penhalonga valley, Umtali, are the ruins of fortifications, the architecture of which is identical with that of the Zimbabye ruins. Again, in driving at water level I have tapped old workings which have been silted up and of which no trace was discernible on the surface, in fact the surface for yards around was covered to a depth of from 4 ft. to 5 ft. with alluvial or rubble, and this is in a perfectly flat country covered with timber, where there was no likelihood of rubble being washed from any distance.

As the author says, the reefs are very irregular and patchy, and perhaps a few examples noted personally may be interesting. On one block of claims with which I am intimately acquainted, the reef runs E and W; at the W end the reef dips N, but at the E end it dips S, whilst about the centre of the claims and at the W boundary two other reefs intersect and cross it, also carrying gold.

Examples 1, 2 and 3 will give some idea of the extent and value of the payable shoots on this block of claims:—

Distance in feet.	Value per ton.	Distance in feet.	Value per ton.	Distance in feet.	Value per ton.
	dwt.		dwt.		dwt.
10	nil	10	10	10	10
20	10	20	15	20	10
30	14	30	15	30	12
40	16	40	15	40	15
50	20	50	20	50	20
60	20	60	20	60	20
70	20	70	20	70	20
80	20	80	20	80	10
90	20	90	14	90	20
100	20	100	14	100	20
114	colour	110	10	110	18
		120	colour	120	colour
				130	14
				140	nil

These samples were taken from drives at water level, over an average width of 24 in. and are pannings; assay values were much higher, in some cases more than treble. It will be seen that the longest shoot was only 130 ft. in length.

The meeting then closed.

THE CAPACITY OF CIRCULAR VATS PER FOOT OF DEPTH

1 TON = 2,000 LBS. By W. A. CALDECOTT. SP. GR. OF DRY SLIMES = 2·7

Diameter of vat in feet.	Circumference of vat in feet.	Cubic feet per foot depth, or Area of vat in square feet.	TONS OF DRY SAND PER FOOT DEPTH.			TONS OF SOLUTION PER FOOT DEPTH.	SLIME-PULP.													Ratio of Dry Slimes to Solution.
			COLLECTED.	LEACHED.	TRANSFERRED.		30% Moisture	35% Moisture	40% Moisture	45% Moisture	50% Moisture	1 to 1½	1 to 2	1 to 2½	1 to 3	1 to 3½	1 to 4	1 to 4½	1 to 5	
							1·788	1·693	1·607	1·530	1·459	1·337	1·266	1·219	1·187	1·163	1·144	1·129	1·117	Sp. Gr. of Pulp.
			21·5 Cubic feet per ton	24 Cubic feet per ton	26 Cubic feet per ton	32 Cubic feet per ton.	25·56	29·08	33·19	38·01	43·89	59·79	75·75	92·00	107·75	123·62	139·91	156·18	172·20	Cubic feet Pulp per ton Dry Slimes.
							TONS OF DRY SLIMES PER FOOT DEPTH OF PULP.													Diam. of Vat in feet.
5	15·71	19·63	0·913	0·818	0·755	0·614	0·767	0·675	0·591	0·516	0·477	0·328	0·258	0·213	0·182	0·158	0·140	0·125	0·114	5
10	31·42	78·54	3·65	3·27	3·02	2·45	3·07	2·700	2·364	2·064	1·789	1·312	1·034	0·855	0·728	0·633	0·561	0·503	0·456	10
15	47·12	176·71	8·21	7·36	6·80	5·52	6·91	6·075	5·324	4·644	4·031	2·953	2·328	1·924	1·638	1·421	1·263	1·132	1·028	15
20	62·83	314·16	14·61	13·09	12·08	9·82	12·28	10·80	9·466	8·256	7·163	5·251	4·132	3·422	2·913	2·535	2·240	2·013	1·827	20
25	78·54	490·87	22·83	20·45	18·88	15·34	19·193	16·88	14·790	12·900	11·192	8·209	6·467	5·347	4·552	3·961	3·509	3·146	2·855	25
30	94·25	706·86	32·87	29·45	27·19	22·09	27·638	24·30	21·298	18·616	16·116	11·813	9·313	7·699	6·555	5·703	5·054	4·530	4·112	30
35	109·96	962·11	44·74	40·09	37·00	30·07	37·618	33·09	28·988	25·284	21·936	16·078	12·676	10·480	8·922	7·763	6·879	6·166	5·597	35
40	125·66	1256·64	58·44	52·36	48·33	39·27	49·13	43·20	37·862	33·024	28·651	21·001	16·556	13·688	11·653	10·140	8·984	8·053	7·310	40
45	141·37	1590·43	73·97	66·27	61·17	49·70	62·185	54·69	48·238	41·796	36·262	26·580	20·954	17·324	14·748	12·833	11·371	10·193	9·252	45
50	157·08	1963·50	91·32	81·81	75·52	61·36	76·77	67·52	59·160	51·601	44·768	32·814	25·870	21·388	18·208	15·844	14·039	12·584	11·423	50
55	172·79	2375·83	110·50	99·00	91·38	74·24	92·90	81·70	71·584	62·437	54·169	39·706	31·302	25·879	22·032	19·171	16·987	15·226	13·821	55
60	188·50	2827·43	131·50	117·81	108·75	88·36	110·56	97·20	85·190	74·305	64·465	47·253	37·252	30·798	26·220	22·815	20·216	18·121	16·449	60
70	219·91	3848·46	178·99	160·35	148·02	120·26	150·49	132·36	115·95	101·14	87·744	64·312	50·704	41·920	35·688	31·052	27·516	24·664	22·388	70
80	251·32	5026·50	233·79	209·43	193·32	157·07	196·55	172·80	151·29	132·196	114·101	83·94	66·349	54·286	46·243	40·212	35·688	32·169	29·153	80
90	282·74	6361·70	295·89	265·07	244·68	198·80	248·76	218·79	191·48	167·312	144·410	106·23	83·974	68·706	58·527	50·893	45·168	40·714	36·897	90
100	314·15	7854·00	365·30	327·25	302·07	245·43	307·11	270·08	236·40	206·560	178·923	131·16	103·672	84·823	72·256	62·832	55·763	50·265	45·553	100

Queries and Replies.

*Concrete Piles in Mills.**—*Reply No. 2.*—Mr. J. H. Frerichs' for the consulting mechanical engineer, Messrs. H. Eckstein & Co., writes stating that the component parts used by their firm in making mortar or cement foundations in their battery houses, are as follows :—One part cement to three parts clean sharp sand and five parts broken stone, to pass through a 2 in. ring. The top 18 in. of foundations contain 1½ parts of cement, the parts of the other ingredients remaining as above.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

THE ESTIMATION OF GRAPHITE.—"When graphite is heated at a bright red in an uncovered dish with ferric oxide several changes may be noted. The carbon entirely disappears, and the ferric oxide, though apparently unchanged at the surface, is darker in colour towards the bottom of the dish, owing to the formation of magnetic oxide. In order to see if the error due to this loss of oxygen could not be avoided, the action of heat and of various chemical reagents on ferric oxide has been studied, until finally it was found that heat alone could be so applied as to solve the difficulty.

If ferric oxide after being heated at a pale red in a covered crucible be then heated in the uncovered crucible at the same temperature, it slowly gains in weight, until after an hour's heating, provided always a fairly even temperature is maintained, the weight remains constant. If the heat be decreased from a pale to a dull red, then the oxide again increases in weight. Slight variations in temperature make practically no difference to the oxide, but it is essential in using the process hereinafter described that as even a temperature as possible should be maintained, and with a little practice this can be done without difficulty, and judging only by the eye. Based on the foregoing observations a process for the estimation of graphite has been devised. The ferric oxide to be used should be heated in a covered crucible at a pale red for two or three hours. The resulting oxide should be a slightly coherent magnetic powder, nearly black in colour. It will be found to be but very slightly hygroscopic. About 5 gm. of this oxide are taken, placed in a small uncovered shallow dish or wide mouth crucible, and heated at a temperature not exceeding a pale red for an hour, with occasional stirring, then weigh. Add about 0·5 of the graphite, stir together, and heat as before for one to two hours, stirring once or twice, and weigh. The carbon will be burnt off, and the oxide remains at the end in the same condition as at the commencement. When one estimation is finished another weight of graphite may be put into the same used oxide. A sample of graphite containing 9·15% of mineral matter by potash fusion gave 8·98% and 8·96% by this iron oxide method. Graphite freed from mineral matter showed by this process 99·73% of carbon.

The mineral matter of coal may be determined in the same way, but the amount found will be about 0·5% higher than as determined in the ordinary manner. This is what might be expected, as when coal is burnt to an ash the constituents are likely to react together, thus giving a figure somewhat lower than the true amount. Individual determinations, whether of graphite or coal, by this process will be found to agree closely; thus, using quantities from 0·5 to 1 gm. of coal, 9·21, 9·24, 9·34, and 9·35% of mineral matter were indicated."—F. BROWNE, F.I.C.—*Chemical News*, July 31, 1908, p. 51. (E. H. C.)

THE RUSTING OF IRON.—"An account was given of a series of experiments from which the following conclusions are drawn.

1. Oxygen or air together with liquid water are alone necessary to produce rusting without the assistance of carbonic acid, failure on the part of other experimenters to produce rusting in iron prepared by treatment with chromic acid being probably due to 'passivity.'

2. Water alone in the absence of oxygen attacks iron slowly, producing a film of what is probably ferrous hydroxide, as it is wholly converted into brown rust on admission of pure oxygen. In this connection, the experiments of Whitney (*Journ. Am. Chem. Soc.*, 1993, xxv., 298), repeated and approved by Walker, Cederholm, and Bent (*Journ. Am. Chem. Soc.*, 1907, xxix., 1255), were confirmed.

3. Iron rust even when very old and constantly exposed to the weather always contains ferrous oxide.

4. The process of rusting is due, in the first instance, to electrolytic action promoted in all ordinary cases by the existence of carbonic acid in water exposed to the air, and by the presence in iron of various compounds of carbon, silicon, phosphorus, and sulphur. That electrolytic action does occur locally in all commercial iron, whether malleable iron, grey or white cast-iron, or steel immersed in water, is shown by the use of the 'ferroxyl' indicator (phenolphthalein and ferricyanide) introduced by Walker and his colleagues, and in other ways. In ordinary rusting the process is also assisted by the depolarising action of the layer of ferric oxide or hydroxide formed externally on the rust spot.

The conditions of rusting of pure iron have never yet been studied, inasmuch as a pure iron in a massive form, if indeed in any form, is unknown."—W. A. TILDEN.—*Chemical News*, July 10, 1908, pp. 20—21. (E. H. C.)

THE ACTION OF HYDROGEN SULPHIDE ON ALKALINE SOLUTIONS OF ZINC SALTS.—"The fact that the zinc sulphide, or the zinc hydrosulphide, precipitated from alkaline solutions of zinc salts by sodium or potassium hydrosulphide is soluble in an excess of these reagents, and that the zinc sulphide, or zinc hydrosulphide, precipitated from alkaline solutions of the metal by hydrogen sulphide dissolves when the gas is permitted to act on the solutions for some time, appears to have escaped the notice of the analytical chemist.

NOTE.—I noticed this peculiar behaviour of zinc sulphide some three or four years ago, and supposed that my observation was a new one. I found, however, that the reaction was first observed by Julius Thomsen in 1878 (*Ber.*, xi., 2044) and subsequently examined by A. Villiers (*Comptes Rendus*, cxx, 97). Lottermoser (*Samlung chem. u. tech. Vortrage*, vi.) and Winsinger (*Bull. de l'Acad. des Sciences de Bruxelles* [2], xiv., 321) also refer to it.

---

* See this *Journal*, vol. ix, No. 3., Sept. 1908, page 98.

At all events, in no work on analytical chemistry to which I have access is this remarkable behaviour of zinc sulphide referred to. The solution of the zinc sulphide is a colloidal one, for the zinc in it will not pass through parchment paper. The zinc sulphide, or zinc hydrosulphide, acts towards sodium and potassium hydrosulphides in much the same way that zinc oxide, or zinc hydroxide, acts towards sodium and potassium hydroxides. The analogy between the two reactions almost compels one to conclude that the change takes place in the sense of the equation $Zn(SH)_2 + 2RSH = R_2ZnS_2 + 2H_2S$.

If, however, an alkali sulphozincate is actually formed, it must be very unstable, for concentrated solutions of mineral salts, when added to its solution, precipitate only zinc sulphide, or possibly zinc hydrosulphide; and then, too, the zinc sulphide separates out gradually, and in a slimy condition, when the solution is allowed to stand.

In many books on qualitative analysis the student is directed to separate manganese from zinc by adding to the solution of their chlorides an excess sodium or potassium hydroxide. Now if the alkaline solution of the zinc, after its separation from the manganous hydroxide, be treated for 15 to 20 minutes with a rapid current of hydrogen sulphide, the zinc sulphide which is first precipitated may dissolve. The smaller the amounts of zinc and sodium chloride present in the solution, and the more rapid current of gas, the more readily does the zinc sulphide formed pass into solution. Should a student pass a rapid current of hydrogen sulphide into such a solution, and then leave the spot and not return until after the lapse of some 15 to 20 minutes, the chances are he will find the solution clear, or nearly so, and report no zinc. I have convinced myself by a number of experiments that there is, in a case of this sort, considerable danger of overlooking the zinc. It is a significant fact that Fresenius uses hydrogen sulphide water in order to test for the zinc in the alkaline solution (*Anleitung z. Qualit. Chem. Analyse*, 1874, 291). He is also careful to state that an excess of the reagent is to be avoided. However, in describing the special reactions of zinc he states that hydrogen sulphide precipitates from alkaline solutions all the zinc in the form of the hydrated sulphide (*loc. cit.*, p. 137). Nothing is said about an excess of the reagent, although he does mention the fact that ammonium chloride greatly promotes the separation of the precipitate.

In the following experiments I used a sodium hydroxide solution of zinc oxide containing 8 gm. of the oxide in a litre:—

1. Ten c.c. of the solution were diluted to 150 c.c. and treated with a rapid current of hydrogen sulphide. The zinc sulphide was precipitated almost immediately, but at the end of 15 minutes it had passed into solution.

2. To 10 c.c. of the solution, made faintly acid with hydrochloric acid, a few drops of a concentrated solution of manganous sulphate were added and the metals precipitated as sulphides with yellow ammonium sulphide. After filtering, dissolving the precipitate in a small amount of very dilute hydrochloric acid, and separating the manganese with excess of sodium hydroxide, the alkaline filtrate was diluted to about 150 c.c. and treated with a rapid current of hydrogen sulphide. In 15 minutes the zinc sulphide had dissolved and the solution was clear.

3. The experiment was repeated with a solution containing the same amount of zinc oxide along with considerable amounts of manganese, cobalt and nickel. The alkaline solution of the zinc oxide, however, behaved toward the sulphuretted hydrogen gas exactly as it did in 1 and 2.

On standing, all three solutions became turbid, owing to a gradual separation of the zinc sulphide." L. W. McCay, *Chemical News. — Journal of the American Chemical Society*, xxx., No. 3, July 24, 1908, p. 40 (E. H. C.)

Detection of Nickel and Cobalt when present together.—"Ammonia is added to the fairly concentrated solution containing the two metals, and then a 10% solution of cane-sugar and dicyandiamidin-sulphate solution (Grossmann's 'nickel reagent'). Finally caustic soda is added. When much nickel is present a yellow or reddish-yellow coloration is produced, while the presence of large quantities of cobalt causes the solution to turn red or reddish-violet. Thus fractions of 1 mgm. of cobalt in presence of more than 100 times the quantity of nickel can be detected by the violet coloration, while if much cobalt and only a little nickel is present, crystalline nickel dicyandiamidin is always precipitated. This test is very accurate and rapid."—Hermann Grossmann and Walter Heilborn, *Berichte der Deutschen Chemischen Gesellschaft. — Chemical News*, July 10, 1908, p. 24. (E. H. C.)

Test for Tellurium. — "The most reliable method for detecting even small traces of tellurium in large quantities of pyrites is as follows:—The pyrites are completely oxidised with strong nitric acid, then evaporated to dryness and absorbed with strong hydrochloric acid. Finally it is boiled, diluted with water, and the insoluble residue filtered. During all the time of filtration and boiling a rapid jet of sulphur dioxide is run through until the filtrate is completely saturated with gas. The tellurium and gold will all be precipitated in the metallic form. These two metals are then filtered and washed with water. The tellurium is then removed from paper filter by washing in an evaporating dish, the water evaporated, strong sulphuric acid added, and gradually heated. When tellurium is present a violet-pink colour appears quickly."—*L'Elettricista*, July 15.—*London Mining Journal*, Sept. 12, 1908, p. 331. (A. R.)

Analysis of Pig Lead.—"Silver, copper and iron are the constituents generally determined in the analysis of pig lead. In certain localities nickel and cobalt are also determined.

Silver is always determined by fire assay. Weigh two 1 assay ton portions of lead, and cupel in a muffle at a low red heat. The cupels should always show feathers. The buttons from the cupellation are weighed and the result reported as ounces per ton.

Copper, iron, nickel and cobalt may all be determined in the same portion of lead.

Weigh 20 gm. of lead into an evaporating dish. Add 80 c.c. of water and 20 c c. of nitric acid, and boil until dissolved. Add 30 c c. of a 50% solution of sulphuric acid, and evaporate to white fumes. Cool and add 150 c.c. of cold water, and boil to dissolve soluble sulphates. Filter and wash by decantation two or three times, using cold water. Transfer the precipitate to the filter and wash four or five times with cold water. The precipitate, which is lead sulphate, is discarded.

Heat the filtrate to boiling, remove from the hot plate, and pass a rapid stream of hydrogen sulphide through it. Allow to settle and filter. Wash the precipitated copper and silver sulphides with hydrogen sulphide water. Dissolve the copper sulphide in hydrochloric acid, allowing the solution to run through the filter.

Wash with hot water. Heat the filtrate, and pass hydrogen sulphide through it to precipitate the copper. Filter and wash the precipitated copper sulphide with hydrogen sulphide water. Burn in a porcelain crucible, and weigh as copper oxide. This weight multiplied by 0·797 will give the weight of metallic copper.

Boil the filtrate from the first precipitation, and add an excess of ammonia. Filter off the precipitated ferric hydrate, and wash with hot water. Burn the precipitate, and weigh as ferric oxide. To find the amount of metallic iron present multiply by 0·7.

Make the filtrate from the iron precipitation acid with acetic acid, and pass a current of hydrogen sulphide through it. Treat the precipitated nickel and cobalt sulphides with yellow ammonium sulphide and dilute potassium cyanide. The nickel sulphide will go into solution and leave the cobalt sulphide floating in the liquid. Filter and wash with hydrogen sulphide water. Burn the precipitate and weigh as cobalt oxide. Multiply the weight of the precipitate by 0·78667 to get the weight of metallic cobalt.

Make the filtrate acid with hydrochloric or sulphuric acid, using only a slight excess. The nickel will be precipitated as nickel cyanide. Filter, wash, ignite and weigh as nickel oxide. This weight multiplied by 0·78667 will give the weight of metallic nickel." — EVANS W. BUSKETT. — *Mining World*, May 2, 1908, p. 709. (H. A. W.)

EXTRACTION OF ARGON. — "Argon can be extracted from the air by simultaneously absorbing the oxygen and nitrogen with calcium carbide. The finely crushed carbide, mixed with 10% of calcium chloride and heated to about 800° C., reacts as follows:—

$$CaC_2 + N_2 = CN_2Ca + C.$$
$$CaC_2 + O = CaO + 2C.$$

The calcium chloride decreases temperature of the reaction. As can be seen from these equations, the nitrogen gives calcic cyanamide ($N = C - N = Ca$), and the oxygen transforms the carbide into a mixture of lime and carbon. Messrs. Fischer and Ringe (*Bericht. der Chem. Gesells.*, 1908, p. 2,017) place the carbide and calcium chloride in a cylindrical iron mould, 3 mm. thick, 50 cm. deep and 17 in. diameter, heated by gas. The volume of air absorbed is measured by a gas meter. The intake of air is closed before complete transformation of the carbide. The residual gas is made to circulate through the carbide by aid of a mercury pump. With 7 kgm. of calcium carbide, 11 litres of argon can be obtained."—*London Mining Journal*, Sept. 12, 1908, p. 331. (A. R.)

THE BUILDING OF EXPLOSIVE WORKS.—"This paper dealt with some of the arrangements made to minimise the distinctive effect of explosions in explosives factories. The author proposed that buildings in explosive works should be constructed in ferro-concrete with fine river gravel. Such a structure being practically a solid mass would not be liable to collapse, it would be fire and lightning proof, and, should an explosion take place, would be so pulverised that particles could not be projected to any considerable distance. In order to resist the fall of heavy pieces of machinery or other ponderous materials, the roof is made with a double ferro-concrete skin, a layer of sand about 1 ft. thick being interposed. The use of wire-glass is recommended to avoid the dangerous splintering of ordinary window-panes."—O. GUTTMANN.—*Times Engineering Supplement*, June, 1908. (J. A. W.)

SMOKE PREVENTION.—"The Brynkinalt Collieries, Ruabon, have just installed at their works a device, patented by Messrs. Smalley and Newton, and manufactured by Smalley, Rice & Evans, engineers, of Stanhope Street, Liverpool, for smoke prevention. The apparatus consists of a fitting for the furnace door in the form of a conical-shaped funnel, the outer aperture of which can be closed at will by means of small doors or shutters. A small steam jet, specially arranged, is carried into the furnace, and by opening the doors of the apparatus to admit a supply of oxygen over the top of the coal and forcing the fire, all the gases, instead of passing away into the atmosphere, are ignited and made to perform their function, thus effecting a saving in fuel and obviating the creation and escape of smoke. Working in conjunction with the smoke preventer are patent fire-bars, the feature of which is a trough along the entire length on top."—*Times Engineering Supplement*, July 8, 1908, (J. A. W.)

SEPARATION OF SILVER CHLORIDE AND IODIDE.—"Hager in 1871 proposed a method of separating the three silver halides based on the solubility of silver chloride in a boiling 10% solution of ammonium sesquicarbonate in which the iodide is insoluble, whilst only traces of the bromide dissolve. The author now finds that even in the cold (25° C.) 10% ammonium sesquicarbonate dissolves 0·018 gm. of silver bromide per litre and the normal ammonium carbonate a still larger proportion. A mixture of silver chloride and iodide, however, can be very accurately separated by heating at 70—80° C. with a solution of 100 gm. of commercial ammonium carbonate and 20 c.c. of 20% ammonia per litre."—H. BAUBIGNY, *Compt. rend.*, 1908, *146*, 335—336. — *Journal of the Society of Chemical Industry*, March 31, 1908, p. 279. (A. W.)

METALLURGY.

TREATMENT OF CONCENTRATES CARRYING COPPER—"The treatment of concentrates as practised at the Bellevue Gold Mine is a distinct departure from any of the ordinary processes. It appears to be, in fact, quite unique as regards metallurgy of gold and a few notes concerning it may not be without interest. The concentrates have proved themselves very refractory. Cyaniding is out of the question owing to the relatively high proportion of copper they contain, and roasting and chlorinating has been tried and found wanting for reasons, which it would serve no useful purpose to enumerate here. Eventually, the process now in use, of roasting and regrinding without the use of mercury, was evolved and the uniformly good extraction of 90% obtained. The scale formed is relied on to catch the gold, and from this it is recovered by amalgamation in a revolving barrel.

An indication of the work done by the roaster is shown by the following sulphur estimations:—

Green Concentrates—	
Total S as sulphide	36%
Roasted Concentrates—	
S (sulphide)	0·66%
S (sulphate)	1·66%
Total S	2·32%

Satisfactory extractions have, however, been obtained with less complete roasts than the foregoing, as much as 3% or 4%, S, as sulphides, not materially altering results. In fact it has been noticed that, if anything, the extraction is improved when the roast is slightly on the green side.

Owing to the highly corrosive nature of the mine water, especially when mixed with the soluble contents of the roasted concentrates, the grinding pan has to be protected. The sides are lined with C.I. plates, 1 in. thick, set in cement, and the centre casting is covered with a C.I. sleeve to prevent its corrosion.

The muller plate and liners last about three months; the lives of all parts of the pan being very indefinite, as the corrosive action is not always uniform in effect.

The scale formed in the pan averages in value about 260 oz. per ton, and accounts for 80% of the extraction. The overflow from the pan is passed over a Wilfley table, which serves a dual purpose. In the first place it separates out any unroasted concentrate, which, though never assuming deleterious proportions, is always present to a greater or lesser extent, according to the perfection of the roast; and secondly, the incrustation, which forms on the riffles, catches a considerable quantity of gold, its assay value averaging 51 oz. per ton. Finally, the tails are passed through a launder, 300 ft. in length, which is filled with old drill steel. The launder becomes gradually full of incrustation, and is cleaned up at three-monthly intervals, the scale averaging 9 oz. per ton, and together with the Wilfley table, accounts for a further 10% extraction, bringing the total up to 90%. The launder product is shipped to smelters. The grinding pan, in which the greater part of the gold is caught, is cleaned up every half-month, or oftener, should assays show any falling off in the extraction. The brown rust-coloured scale, which covers all parts of the pan that are submerged, varies somewhat in composition and appearance. Sometimes it is hard and brittle, necessitating chipping with sharp hammers, and at other times it is quite soft and can be scraped off. The cause of this difference is not quite clear, but is apparently due to variations in the quantities of soluble copper in the water (the soft scale is invariably poorer in gold and richer in copper than the hard scale). Strictly speaking, two kinds of scale form simultaneously on the pan. Besides the brown scale referred to, which often attains a thickness of a ¼ in., there is between this and the iron a thin layer of lustrous black scale, but analysis of the two show that the latter is probably only the residual matter from the decomposition of the cast iron, and performs no direct function in recovering gold.

The Brown Gold-Bearing Scale—

Cu (metallic)	57·25%
Fe_2O_3	26·72%
FeS_2	3·69%
$CaSO_4$	10·76%
Au	0·92%
	99·34%

The Black Scale next to the Iron—

Fe_2O_3	61·38%
FeS_2	10·87%
Cu	1·60%
C (graphite)	12·70%
SiO_2	12·60%
	99·15%

Gold, 4 oz. 10 dwt. per ton.

The incrustation on the Wilfley table is very similar to the above, but contains considerably less copper and more $CaSO_4$. It is essentially a gypsum incrustation.

Clean-up.—The pan is chipped and scraped clean of all the adhering scale. The muller is then lifted, the dies removed, all the material in the interstices collected, and everything taken direct to the amalgamating barrel, where it is ground with the requisite quantity of mercury for 36 hours, fully that time being necessary to gain the maximum amalgamation. The amalgam recovered is very base, consisting mostly of copper, and after retorting assays about 10 to 12% gold, but this varies within wide limits. The pannings from the barrel assay 12 oz. per ton, and are shipped to smelters periodically. The retorted material is smelted with sulphur, the resulting mattes being run down with iron till too poor for further treatment in this way. The base bullion is refined with sulphur, the matte resulting from this being, of course, again run down with iron. The ultimate products are bullion worth £3 12s. per oz. and matte assaying 10 oz. per ton. The latter is shipped.

The Principles Governing the Process.—In seeking an explanation for the recovery of gold by this process we are confronted at the outset with the main possibilities.

(1) *Chemical Reaction.* — The only argument in favour of this possibility is the high percentage of the extraction and the high value of the scale, which seems beyond the bounds of mechanical concentration. On the other hand, against the chemical reaction hypothesis are several facts which preclude its acceptance. If the gold is first dissolved and then re-precipitated on the iron, one would naturally expect to find it in a very finely divided state, similar, for instance, to that deposited on zinc in the cyanide process; but, on the contrary, the gold in the scale is comparatively coarse and flaky, a fact which can be clearly seen by treating some of the scale with nitric acid and dissolving out the copper, and moreover it resembles in appearance the gold obtained by panning off the roasted concentrates. Again, only the slightest traces of gold can be found in a filtered sample of the water taken from the pan.

(2) *Mechanical Concentration.*—Before going into the matter of mechanical concentration in the pan, there are one or two facts, bearing indirectly on the subject, that are worth noting. The first of these is the coarseness of the gold in the roasted concentrates. The following grading test on a rich sample illustrates this clearly :—

	%	oz.	dwt.
Remaining on 60 mesh	22·75,	4	5
Through 60 and remaining on 80 mesh	8·50,	3	14
,, 80 ,, 120 ,,	35·00,	2	16
,, 120 ,, ,,	33·75,	1	14

The material remaining on the 60 mesh on regrinding graded as follows :—

Through 120 mesh	97,	2	0
Remaining on 120 mesh	3,	74	0

Now by ocular examination of the gold obtained by panning off some of the roasted concentrates it is quite evident that the particles are thin and flaky, in fact so much so that some of the larger particles can only be tailed with great difficulty.

On the Bellevue mine at the present time, accumulated tailings are being treated by concentration on Wilfley tables, with subsequent roasting and regrinding of the concentrates by the method previously described. After the Wilfley tables have been in use a few days, an incrustation begins to make its appearance on the riffles. This is practically a gypsum growth, crystals of calcium sulphate being deposited from the very dense lake water used in the plant. After a few weeks the incrustation becomes so thick as to impede the action of the table and it is then scraped off. The average assay value is 38 oz. per ton and the gold is recovered by roasting and amalgamation, adding very materially to the monthly output of the mine. By drying, crushing, and panning off, the gold in the scale presents the same flaky appearance, which has been already referred to.

Imagine the growth of the incrustation step by step. First, on the wooden riffles and linoleum a layer of crystals is deposited, with their longer axes projecting at right angles to the surface they rest on as is the habit of gypsum, and thus forming a natural rough surface on which the gold particles are caught, together with particles of mineral, gangue, etc., but not so much of these latter, as they do not by reason of their shape lend themselves so readily to catchment as does the gold with its flat jagged form. Deposition of $CaSO_4$ is going on continually and consequently fresh rough surfaces are always being exposed to the passing stream. At the same time the gold already caught in the interstices is covered over and secured.

Reverting now to the treatment of the roasted concentrates it has already been pointed out that the gold exists in this peculiar form that is very apt to get caught on a rough surface, and a further confirmation of the fact exists in the formation of a very rich gypsum incrustation in the wetting down launder, similar in most respects to that described in connection with the tailings plant. So far, then, the high values of the incrustation and scales have been accounted for, but hardly sufficient evidence has been adduced to explain the high percentage of gold recovered.

This brings us to what is probably the chief underlying principle of the whole process. The concentrates consist chiefly of pyrites and pyrrhotite, the former carrying copper. A sample of the pyrites showed 1% Cu on assaying. On roasting a quantity of this copper is converted to the sulphate. An analysis of the constituents of the roasted concentrates that are soluble in water is as follows:—

CuO	1·36%
CaO	0·64%
MgO	0·28%
FeO	1·45%
CoO	0·13%
SO_3	0·395%
Cl	0·16%
Na and K	Undetermined

These figures were determined by agitating a weighed sample of the roasted concentrate with distilled water; then analysing the filtered water. Note the copper and calcium.

In actual treatment the concentrate is wetted down with about six to eight times its weight of mine water. An analysis of the filtered water from this mixture gave:—

CuO	0·17%
CaO	0·38%
FeO	0·07%
Cl	8·60%
SO_3	0·76%
Mg Co, Na and K	Undetermined

Note the presence of the soluble copper salt and also the calcium contents chiefly present as $CaSO_4$ in the mine water, which is very dense and always shows a tendency to form an incrustation on the vessel in which it is contained.

The significance of the presence of copper in the pan water and as the chief constituent of the scale (see analysis) is at once apparent, but the exact action can only be a matter of close surmise. On entering the pan the dissolved copper sulphate reacts with the iron forming ferrous sulphate and cement copper.

$$CuSO_4 + Fe = FeSO_4 + Cu.$$

Each successive molecule of copper liberated is deposited not on the iron, but on another molecule of copper which is in electrical connection with it. If a piece of gold be in contact with iron in a solution of $CuSO_4$ a film of copper is deposited on the gold too. The following experiment illustrates the above fact and also the way in which the gold may be embedded in the scale.

A sample of the gold bearing scale was pulverised, treated with HNO_3 to remove the copper and then panned off. The gold particles, freed from impurities, were transferred to a clean beaker and a solution of copper sulphate added. A clean flat piece of iron was then introduced and stirred round amongst the gold. In a few minutes practically all the gold had adhered to the rod, being securely embedded in the spongy copper which, of course, had been precipitated on the iron. Now in the grinding pan most of the copper would be deposited on the wearing surface of the shoes and dies which are always kept bright by the continual grinding. Here, too, a considerable portion of the gold is freed from its encasing gangue and it is not unlikely under these circumstances that it should become embedded in spongy copper as it was in the experiment. On being forced up again into the body of the pan and swirled round, the particles so covered on coming in contact with the film of copper on the side of the pan would adhere tenaciously. In this manner the scale would grow, but being of a soft and spongy nature, it would be soon rubbed off by the swirl in the pan, were it not for the cementing and hardening action of the calcium sulphate in the mine water, which in precisely the same way as described in connection with the tailings plant, binds the scale into a tough mass completely securing the gold. It is on record that on one occasion the pan was run on fresh water for a few days, with the result that the extraction was lowered considerably, as might be expected in the absence of the cementing agent, $CaSO_4$. This confirms to some extent the argument concerning the binding action of the mine water.

This practically completes the remarks in explanation of the mechanical concentration theory, but whether the case as put is strong enough to account for catching 80% of the gold in the pan may be a matter for discussion, but looking at it broadly, the coarseness of the gold, its peculiar shape and aptness to catch on rough surfaces, the deposition of copper in the pan and high percentage of copper in the scale, everything points to this being the correct solution of the problem. The gold caught on the Wilfley tables and in the 300 ft. launder are practically applications of the same principle exerted in the pan, though, of course, coming after the pan has had the first selection the resulting scales are not nearly so rich in gold."—B. L. GARDINER, B.Sc.—*Monthly Journal of Chamber of Mines of Western Australia*, May, 1908, p. 181. (H. A. W.)

ACCURACY OF COAL ANALYSES.—"From experiments it has been found that an increase of water gives an increase in the volatile matter. The following figures illustrate this point. In the first column is the analysis of the original coal, in the second the coal plus an addition of 5% water, in the third with an addition of 12% water. The results are calulated to the pure coal (*i.e.*, less moisture and ash):—

Volatile matter	37·02	37·86	38·44
Fixed carbonaceous residue ...	62·98	62·14	61·56

The presence in the coal of carbonates also affects the results. In a coal which contained 12% carbonic acid (present in the coal as carbonate of lime and magnesia), recently analysed, the amount of cabonic acid left in the coke after the estimation of volatile matter was only 0·9%, hence 11·1% had gone off as volatile matter. Further, it is not possible to say to what extent the reaction $CO_2 + C = 2CO$ goes on, but this must have a considerable effect on the result. Pyrites in the coal must also have some effect, but it is not easy to judge to what extent. It will be readily seen from the above remarks that the proximate analysis is not sufficiently reliable as a basis for purposes of classification.

In ultimate analysis, the way in which moisture is estimated may affect the percentage of practically all the constituents when expressed in percentage of the pure coal. In one example the difference between moisture obtained by drying in the toluene bath and over sulphric acid *in vacuo* for twenty-four hours was 0·3%. It is exceptional to find so big a difference in moisture by the two methods, but where coals rich in moisture are being dealt with it is better to estimate by both methods, as an idea of the possible error in composition is obtained.

In estimating ash, the value for the ash left in the combustion-tube is almost invariably higher than that obtained by ashing in the muffle. The reason appears to be that in the combustion-tube, where the coal is burned in oxygen, more of the sulphur is converted to sulphuric anhydride, which combines with any lime in the ash, whilst in the muffle the atmosphere is less highly oxidising, so that more of the sulphur goes off as a lower oxide, instead of combining with the lime in the ash. In every case where the ash has been estimated in both ways the muffle has given the lower result both in ash and sulphur-in-ash. Should minerals containing ferrous compounds be contained in the coal, these will, on combusting the coal in oxygen, become (to a great extent at any rate) ferric. Hence the ash, as found by analysis, will be greater than the original ash in the coal, and the oxygen which is obtained by difference will, consequently, be too low.

When coal containing pyrites is burned in oxygen, the pyrites is converted into ferric oxide and oxides of sulphur; the latter are absorbed by the lead chromate in the combustion-tube, while the former remains in the boat and is weighed with the rest of the ash. The pyrites cannot be regarded as part of the organic combustible constituents of the coal any more than the rest of the ash and the moisture, but for every 240 parts of weight of pyrites present in the coal 160 parts of the ferric oxide and 128 of sulphur are being counted. The effect of this is that the oxygen, which is obtained by difference, is too low.

Carbonates in a coal are more or less decomposed during analysis and give off carbonic acid gas. This is absorbed by the potash-bulbs and is weighed with the carbonic acid formed by the combustion of the carbon of the coal, the result being that the carbon is too high, the true ash too low, and the oxygen too high. When the results of the analysis are calculated on the pure coal all constituents will be affected. In practice it has been found that the amount of CO_2 left in the ash varies considerably with each estimation, so that to make the necessary correction the CO_2 has to be estimated specially in the ash left after each combustion. When this is done duplicates agree well.

Where both carbonates and pyrites are present there will naturally be complications, as the oxides of sulphur will combine with the lime or magnesia of the carbonates, thus lowering the combustible sulphur and increasing the sulphur-in-ash. It is not possible, however, to correct for this.

The following are, briefly, the conclusions drawn from Professor Fischer's researches:—Coals contain varying quantities of unsaturated compounds which rapidly absorb oxygen, thereby gaining in weight but deteriorating in coking properties and calorific value. Another series of compounds also occurs which take up oxygen, but give off carbonic acid and water in the process. The latter process, which is usually slow, produces a loss in both the weight and value of the coal. A coal on storing, therefore, may gain, lose, or remain constant in weight, according to the quantities and relative proportions of the two classes of compounds present, but will almost invariably deteriorate in value. When coals are stored in a cool dry place the alteration is, in most cases, inconsiderable. Moisture certainly assists in the oxidation of the coal. The effect of pyrites on spontaneous combustion is undoubtedly over-estimated. The value of ventilating stored coals is doubtful, as although ventilation will help the cooling, it will supply the oxygen necessary to produce combustion.

In order to see to what extent the composition of coals would be affected by keeping, four of the samples stored in the museum of the Geological Survey were recently re-sampled and analysed afresh. The coals have been stored in the basement of the Jermyn Street Museum, where they were not subjected to great changes of temperature, and the samples consisted of (1) anthracite from the Nine-Foot Vein (stored four years); (2) steam coal from the Nine-Foot Vein (stored four years); bituminous coal from the Meynyddislwyn Vein (stored four years); and bituminous coal from the Top Coal of the Rock Vein (stored four years and seven months).

It was noticeable that the anthracite altered least (if at all), the steam coal only slightly, whilst the bituminous coals changed considerably. In each case the change was in the same direction, and though no great accuracy can be claimed for the specific gravity of the pure coal, owing to possible sources of error (to say nothing of the pyrites having partially decomposed in the stored specimen), yet the general indication was that the greater the alteration in the composition the greater the alteration in the specific gravity. The differences were:—

	(1.)	(2.)	(3.)	(4.)
Loss of carbon ...	·02	·24	1·39	·96
Gain in specific gravity	·007	·013	·026	·020

The change, at any rate in the bituminous coals, is greater than can be reasonably put down to experimental error."—*Iron and Coal Trades Review*, July 10, 1908, p. 139. (A. R.)

MOLYBDENUM.—"Molybdenum is fast becoming recognised as one of the most valuable constituents of a steel alloy. It increases the elongation and elastic limit of steel. In this respect it is much more valuable than nickel. An addition of 0·25% of molybdenum increases elongation from 4% up to 45%. This gives it great value in the manufacture of large crank and propeller shafts. Its introduction into boiler plates for high pressure boilers has made possible the use of lighter plates that are actually stronger than much heavier plates made without the alloy. For motor car forgings and castings molybdenum steel is specially adapted. In high speed tools it gives all the desirable qualities with carbon kept below 1%.

Ferro-molybdenum ranges in composition from 85% to 50% molybdenum and from 3·07 to 0·35% carbon."—*Canadian Mining Review*, May 1, 1908, p. 177. (K. L. G.)

Estimation of High Temperatures. — "In metallurgical operations it is often useful to be able to regulate the heat within a specific limitation. This may, of course, be done by the use of a pyrometer. But in works where it is not necessary to keep such delicate instruments on hand it is sometimes desirable to measure temperatures with a fair degree of accuracy. Many simple methods capable of yielding approximately accurate results have been proposed by various metallurgists. An effective method which has been used for many years with satisfactory results has been developed by Dr. F. W. Skirrow, of Manchester, England. Whenever he desires to carry out a thermal operation at a specific temperature he secures a substance having a definitely ascertained melting point at the desired temperature and another with a melting point a little higher. Each substance he places in a narrow steel tube of any suitable length with one end sealed up. A stiff wire is placed in the tube and one end is fused into the fusible substance at the bottom of the tube while the other end protrudes from the open end of the tube. This wire remains fixed to the substance at the bottom of the tube so long as the temperature remains below its melting point. When, however. the substance melts, the wire becomes loose and can be freely moved up and down the tube.

If a number of such tubes are prepared with numbers or other identification marks stamped on them and containing a series of substances covering the various temperatures within whose melting point range a metallurgist desires to work, very effective work can be done, notwithstanding the cheapness and simplicity of the indicators.

In practice it is found only necessary to use two such tubes in order to keep a fire at a specific temperature. The fireman needs to regulate the firing so that one wire is loose while the other fused in the substance with a slightly higher melting point than the described temperature remains fixed and cannot be moved up and down the tube.

Whenever it is desirable for any reasons to avoid the use of iron or steel tubes, tubes made of other materials may be employed. When tubes made of refractory substances, as earthenware or porcelain, are used, it is necessary to take into account the greater temperature lag of such substances."—*The Engineering and Mining Journal*, June 27, 1908, p. 1295. (W. A. C.)

To Prevent Crystallization of Shafts. — "A device recently patented by F. S. Kirkland, of Milwaukee, Wis., intended to obviate the crystallisation in iron or steel shafts due to repeated concussion such as takes place, for instance, in the case of a cam shaft on a stamp mill. Strip of copper or other non-magnetic metal inlaid into a spiral groove about the shaft. The efficacy of the invention depends upon the assumption that the crystallizing of shafting extends from the outside towards the centre, and by means of the inlaid strip of metal an effort is made to prevent this crystallization by breaking the harmony of the vibration extending through a metal bar. The metal strip is an alloy of copper, zinc, and in some instances of nickel, though this latter is not essential, as the copper and zinc alloy is non-magnetisable. The strips are welded to the bar in the hammering of the iron or steel shaft. They are about ½ in. deep and 1 in. wide, and a perfectly adhesive weld is made."—*Mines and Minerals*, July, 1908. (W. R. D.)

Ignition Points of Wood and Coal. — "*Swedish Timber* (old sleepers from Wigan 4 ft. seam). Approximate age, six years. — This, with various tests, smouldered at 200° C., 180° C., 210° C., 220° C. (average 206°).

Welsh Larch (part of 11 ft. bar from same seam, and about same age).—1st and 2nd tests: smouldered at 360° C. Temperature increased to 400° C., at which point it had not burst into flame.

American Pitch Pine.—From a baulk (about five years) in haulage engine house, Wigan, 4 ft. — Smouldered at 315° C.

Norway Fir.—Old prop out of goaf in Wigan 4 ft. —1st test, smouldered at 260° C., 2nd test, smoked at 280° C., and smouldered at 360° C,. 3rd test, smoked at 260° C., smouldered at 355° C.

Old Timber.—From seat of fire at Pemberton 4 ft., rotten but fairly solid.—1st and 2nd tests smouldered at 230° C., 3rd test (material charred) smouldered at 130° C., 4th test (material softer than in 1st test) smouldered at 160° C., in fireclay muffle, 5th test, decayed timber not out of mine, smouldered at 200° C. in fireclay muffle.

Arley Coal, from colliery near Wigan.—Smoked at 200° C. Coal caked and smoke increased in density at 400° C., when inflammable gas ignited.

Cannel, from Wigan cannel mine.—Smoked at 180° C. Great amount of smoke and gas, which fired at about 400° C.

Pemberton 5 ft. Coal.—Smoked at 140° C., started smouldering at 360° C.

Black Bass, from about 5 yards below 5 ft. (the fire was chiefly in this metal).—Smoked at 170° C. Smoke increased rapidly up to an igniting point of about 400° C.

Six Feet Coal, from 18 in. above Pemberton 5 ft. (This is thrown into the goaf.) Smoked at 160° C.: the smoke increased rapidly to igniting point, 400° C.

The general results of these tests, Mr. Hall continued, show that timber will ignite in an atmosphere at a lower temperature than coal or cannel; and that particularly decayed timber, such as old pit props, ignited most readily of all. It is possible that fires underground would not be so frequent if care were taken to clear out all old or used timber from seams where heating takes place. The action of dry rot on timber seems to make it inflammable at quite low temperatures."—*Iron and Coal Trades Review*, July 10, 1908, p. 150. (A. R.)

The Passing of the Tailings Wheel.—"The tailings wheel, that familiar object on the landscape at the Rand, has long been threatened and is now quietly passing from our midst. For a long time the mechanical engineers of the various groups of mines—notably A. M. Robeson, of the Corner House—were experimenting with various forms of elevators and pumps, but the actual displacement of the tailings wheel can now for the first time be chronicled.

It has been ousted by an improved type of centrifugal pump, evolved after lengthy and careful experiment by the mechanical engineering department of H. Eckstein & Co., to whose initiative Rand metallurgical practice owes so much. The history of the experiments is interesting. When the tailings wheel of the Rose Deep broke down last year a centrifugal pump was used temporarely to replace it. A fatal flaw in the centrifugal pump was found to be the speed with which the manganese steel liners wore out, and also the large waste of power. These were eventually overcome by replacing with locally-made cast white-iron liners, and by intelligent design, with such satisfactory results that the general adoption of this device is now only a question of time. In operating costs the centrifugal pump possesses no advantages over the tailings wheel; but in capital expenditure or prime cost, a marked saving is effected. The cost of a pair of pumps is, roughly, £1,780; the usual cost of a tailings wheel is £8,000. For small equipments the difference in favour of the centrifugal pump becomes more apparent than for larger ones.

The centrifugal pump is notoriously inefficient, but its failings in this respect are greatly exaggerated. With sand the average efficiency is 40 to 45%, and it is hoped to obtain an efficiency of 50%, which nearly approaches that of modern high-lift centrifugal pumps. The centrifugal pump has already displaced the tailings wheel at the Rose Deep and the Crown Deep, at both of which mines the tailings wheel has now been finally 'scrapped.' At the New Heriot a plunger pump has been scrapped in its favour. At the Village Main Reef, one has been erected as a stand-by, and two pumps are being erected at the Glen Deep and two at the Durban Roodepoort Deep. At all the mines that have adopted the pump, duplicates will be erected. It is noteworthy that the life of a tailings wheel averages only 12 years, and many of the wheels on the outcrop mines and first row of 'deeps' are now nearing their end."—*Mining and Scientific Press*, Aug. 22, 1908.—*S.A. Mining Journal*, June 6, 1908. (W. R. D.)

MINING.

SHAFT SINKING IN WET STRATA.—"Mr. L. Morin, engineer-in-chief of the Liévin collieries, has published a note on the cementing of the soil in cutting through the water-bearing level in shaft No. 3. The shaft in question, 6 metres in diameter, is an air shaft, which at a depth of 19 metres touched the water level; the flow of water was considerable, 250 c.m. per hour. Freezing was too expensive, and it was decided to adopt the system already applied at Courrières and Lens. A bottom was constructed at the base of the tubbing already in position, and cement injected underneath the bottom, aiding penetration of the cement into the soil by means of boreholes. The injections and boring had to be made alternately, so that the inrush of water could not exceed the limit for which pumps were available. The temporary bottom was laid March 12, 1906, and the first borehole was sunk May 16, all the work being finished May 25. The inrush of water after removing the temporary bottom was only 4 c.m. per hour. Another bottom had to be constructed a little lower. The work cost 93,000 francs, representing an economy of about 120,000 francs compared with the freezing method."—*Le Ciment*, Aug. 1908.—*London Mining Journal*, Sept. 12, 1908, p. 331. (A.R.)

ON CERTAIN ERRORS IN COMPUTING ORE VALUES.—"These errors may be due to the failure to take into account the more or less regular distribution of the ore in the plane of the vein comprised within a particular block to be estimated.

Distribution of Ores.—In order to study these relations the form of distribution may be reduced to two types. According to the first the pay ore is assumed to occur in regular shoots or streaks; according to the second type, the ore is assumed to occur in patches, the form of which for the sake of discussion will be assumed as square. The width of the vein will be taken as unity and the block of ore to be investigated is supposed to be a square, limited, say, by two levels and two winzes. In Figs. I. and II. the shaded portions represent a high-grade ore of homogeneous value, and the blank portions low-grade ore, also of constant value In Figs. III. and IV. the shaded portions denote ore and the blank portions barren gangue. The values are thus supposed to be distributed in regular geometrical form uniformly through the block and it is granted that the samples drawn at regular intervals from all sides of the block represent the average value of the exposed faces.

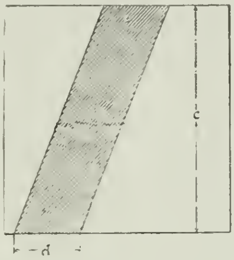


FIG. I.

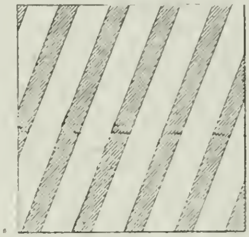


FIG. II

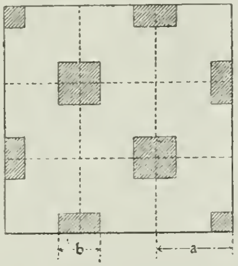


FIG. III.

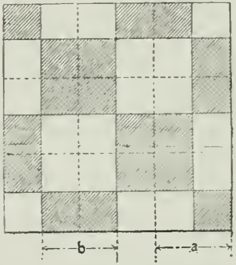


FIG. IV

Under these conditions it may be of interest to compare the values resulting from an average of the assays drawn from the perimeter of the block and the values drawn from the mill returns when this block of ore shall have been stoped out and treated. The first quantity may best be expressed by L = average assay of the perimeter and the second quantity by S = the average assay of the block after extraction.

Ore in Regular Shoots.—Fig. I. represents a block in which a high-grade ore occurs as a simple pipe, or chimney, the length of which in the drifts is represented by d. The rest of the block is occupied by a low-grade ore; c = the length of each side of the black; m = assay value of the high-grade ore; n = assay value of the low-grade ore; L = average value per ton derived from samples drawn at equal intervals from all the exposed faces of the block; S = the actual value per ton of the block as a whole, hence

Computing Ore Values.—

$$L = \frac{2md + n(4c - 2d)}{4c} = \frac{md + n(2c - d)}{2c}$$

$$S = \frac{mcd + n(c^2 - cd)}{c^2} = \frac{md + n(c - d)}{c}$$

Take as an example a block 50 ft. square with a high-grade shoot measuring 30 ft. along the drift. Let $m=\$20$ and $n=\$5$, then

$$L=\frac{20\times 30+5(100-30)}{100}=\$9.50$$

$$S=\frac{20\times 30+5(50-30)}{50}=\$14.00$$

The discrepancy in the results varies with the proportion between the high and low-grade ore and with the difference between the values of the two grades until when $n-0$, $L:S=1:2$.

This, of course, is a case so simple that the relations would be immediately apparent to the engineer and duly discounted. But the relations would be less obvious if instead of the ideal case considered, the ore occurred in several chimneys. It is, however, plain that the greater the number of streaks in one block, the less will be the error until L and S become practically equal in Fig. II.

Ore in Bunches or Patches.—The second type of occurrence to be considered assumes the ore to be distributed throughout the vein in bunches or patches which, reduced to the simplest form, may be regarded as squares placed at regular intervals. According to Fig. III. a represents a side of each square which determines the distribution of the patches, and b represents a side of each square patch of ore. The average assay then of samples drawn from all sides of the block,

$$L=\frac{6mb}{12a}=\frac{mb}{2a}$$

whereas the average assay of the whole block,

$$S=\frac{4\cdot 5mb^2}{9a^2}=\frac{mb^2}{2a^2}$$

The ratio then of $L:S=a:b$. From this it appears that when the patches are small in proportion to the area of the block, the average assay drawn from the exposed faces will be relatively too high and will only approximate accuracy when b approaches the dimension of a. When $b=a$, the area occupied by the ore equals the area occupied by barren gangue as represented in Fig. IV., and when this proportion is exceeded, that is, when the distribution may be described as patches of gangue in a field of ore, then the average value of the samples drawn from the perimeter yields an underestimate of the value of the block."—Henry Hobart Knox.—*Engineering and Mining Journal*, April 18, 1908, p. 806. (G. H. S.)

Safety Sinking Hooks.—"In shaft sinking it is the usual practice to work with two buckets, which are alternately attached to the winding-rope, so that one may be filled at the shaft-bottom, while the other is being emptied at the surface. It was through an accident that I was prompted to devise the hook illustrated in Fig. I.

A sliding ferrule, a, fitted to the tongue of the hook, is prevented from dropping off it by a pin passing through a slot c in the tongue d. To open the hook, the ferrule is simply pushed upward as far as it will go, and when the hook is closed, the ferrule slides down over the point b, thus firmly locking the hook, which cannot be opened without sliding the ferrule up. The safety of the hook is thus independent of the spring, which may, indeed, be dispensed with if desired, although I prefer to retain it. Several of these locking hooks have been made and used and are found to answer perfectly the object for which they were designed.

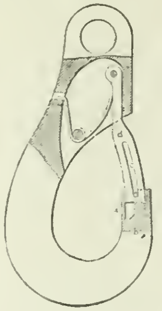


Fig. I., Safety Hook.

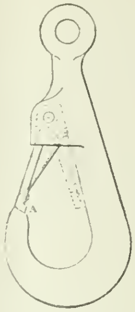


Fig. II., Spring Hook.

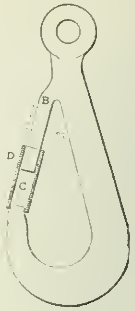


Fig. III., Safety Hook.

For many years sliding ferrule sinking hooks have been satisfactorily used at the Baggeridge colliery, in South Staffordshire. Continual trouble, however, was experienced by the breakage of the ordinary spring, or by its losing temper and becoming weak; this fault was overcome by arranging the spring as shown in Fig. II. The small projection A prevented the spring from being dashed against the back of the hook, but it did not remove the disadvantage caused by the loss of elasticity in the steel. In order to prevent any possibility of the bow of the bucket dropping off the hook, a machinist at the Baggeridge colliery, arranged for the hinge B, of the spring-portion to be welded to the top, as shown in Fig. III., leaving a sufficient space C, between the point of the hook and the slide-piece D, for the bow to pass through: this was sealed during all ordinary operations by the sliding ferrule D which always dropped unless it was deliberately propped up. Experience proved that the hook was more safe, when the spring was entirely dispensed with."—Henry Louis.—*Engineering and Mining Journal*, April 18, 1908. p. 817. (G. H. S.)

MISCELLANEOUS.

The Ventilation of Factories. "The removal of dust and fumes, often a most important and difficult problem in factory ventilation, was the subject of the fourth Shaw Lecture on Industrial Hygiene, delivered by Dr. J. S. Haldane before the Royal Society of Arts, on February 27, and published in the Journal for May 22. This is a subject on which Dr. Haldane can speak with authority, on account of his connection with the work of the late Home Office Committee on Factory Ventilation. It is impossible here to reproduce the many practical illustrations he gives of the principles laid down in his paper, but the following abstract giving the main points of his general review of principles and methods will be found of interest.

Whether or not any variety of dust is known to cause dangerous effects when habitually inhaled, I think that every kind of dust produced in manufacturing processes ought, as far as practicable, to be prevented, or removed from the atmosphere in which the work people are present. The reason for this is not only that dusty air is, at the best, unpleasant to breathe, but that when dust is present the clothes, skin and hair become very dirty, untidy, and uncomfortable. This inevitably tends to lower the social status and self-respect of work people, if, at any rate, they have to go back to their homes in the same untidy condition. When dust and dirt cannot be avoided the provision of overalls, or of means of washing and changing clothes on leaving work, is extremely desirable. . . .

Apart from the reason which I have just referred to, it is often urgently necessary, on the score of health, to prevent the inhalation of dust. Certain kinds of dust, when constantly inhaled during work, produce in time most disastrous effects, and we may pretty confidently say of every kind of dust that it is to some extent harmful.

Of dusts which are definitely harmful, a very important class is that from the disintegration of hard stone or other material. The dust produced in mining hard rock or in stone dressing, the flint dust used in the pottery trade, the dust from dry grinding in the metal trades, and the steel dust produced in file cutting and other work are most harmful in their effects. Striking evidence of the disastrous effects of the inhalation of hard rock dust is given in a table showing the death rate of Cornish miners, 94% of whom die of lung disease at an average age of 37. Other dangerous dusts are those from poisonous materials of any kind, particularly from lead compounds. Dusts from textile materials are less serious in their effects, and coal and cement seem to have but little influence on the health of workers. Fumes from chemical processes and poisonous gases are always dangerous and should not be allowed to escape into the room at all.

In many cases the best way of dealing with dust is to prevent its formation altogether. This can be effected by substituting wet for dry processes, and fortunately much of the most dangerous dust can be dealt with in this way—in particular the dust from disintegration of hard stone or steel. Thus the use of a jet of water prevents dust formation in rock-drill and other work in tin and granite mining, special rules to this effect being now in force. It is to be hoped that in all cases where dust from disintegration of hard stone, fire-clay bricks, and similar material, is apt to be inhaled, wet methods will also be adopted where possible. The substitution of wet for dry grinding, and for dressing of grindstones is another important step in the right direction, and I have little doubt that in many other dusty processes it would be practicable to use wet methods, though, unfortunately, wet processes are quite out of the question in very many cases.

When dust formation cannot be avoided, its escape can sometimes be prevented by entirely boxing in the dusty process. Where the dust is itself the product of the process, as in the grinding or breaking up of material, efficient boxing in is an advantage to the process itself as well as to the persons employed in it. Where the dust is in other ways of some value, the same consideration applies. The use of dust-proof arrangements for filling and conveying dusty material, particularly where the dust is poisonous or otherwise dangerous, is a further advance in the same direction. By means of simple mechanism for this purpose much dust inhalation and loss of material may be prevented.

In processes where fumes or noxious vapours are emitted closed vessels ought also to be used wherever possible, and if a closed vessel, or a boxed in machine, is also connected with an exhaust pipe, the fumes and dust are prevented from escaping at times when the vessel or machine has to be opened.

In most cases it is unfortunately not possible either to prevent the formation of dust or to box in the dusty process completely, and the only method available if to draw away the dust by means of an air current. There are certain general principles applicable to the removal of dust in this way. In the first place the dust ought to be removed at, or as near as possible to, the point of origin. The advantages of this are evident; by this means the dust is prevented from getting into the general atmosphere of the room and being inhaled by those present, as well as settling everywhere. A far smaller volume of air is also sufficient to remove the dust. This is important, not only from the point of view of expense, but because draughts and cold are also prevented. When dust is permitted to get into the general atmosphere of a factory, enormous volumes of air are required to carry it away, which means that equal volumes have to come in from outside, so that warming or prevention of draughts may be quite impracticable.

A second general principle is that the air current from the source of dust to the exhaust opening should, as far as possible, envelop the source of dust, and be of sufficient velocity to carry the dust with it in spite of the ordinary slight draughts existing in the room, or produced by the dusty machine. It is unfortunate that in whatever direction an exhaust opening may point, the air entering is sucked in from all sides. Hence the linear velocity of the draught towards the opening diminishes very rapidly with increasing distance; and at a distance of 2 or 3 ft. an exhaust opening, unless very large, will fail to carry off dust efficiently, particularly from a machine which itself causes draughts. . . . An air current from an inlet can be directed from it, owing to the momentum of the moving air, but not a current to an outlet. The source of dust has therefore to be enclosed, as far as possible, by hoods or other coverings, to serve as air guides. When a dusty machine can be enclosed on all sides, except where the material enters or leaves it, a very satisfactory result can be obtained. Where such enclosure is not possible, the best that can be done is so to arrange the inlets and outlets of air to the room that the dust is on the whole carried directly towards the outlets. A further important principle is, that as all dust tends to fall, it is often best to remove it in a downward direction. This applies particularly in cases where the air current carrying the dust towards the exhaust opening is very slow.

It must not be forgotten, finally, that when exhaust ventilation is used for removing dust, provision must be made for corresponding inlet ventilation to the room, the air being warmed, if necessary. It is not uncommon to see exhaust ventilation rendered partially inoperative by failure to provide proper inlets.

In producing an air current along a duct we are imparting motion, and therefore energy, to a large quantity of gas. The amount of energy thus imparted—in other words, the work done upon the air—varies in proportion to the mass of air moved and the square of its velocity. In the case of an air current in a duct, however, the mass moved varies directly as the velocity. Hence the total work done will vary as the cube of the velocity of the air current, measured at any one point in the duct. The work done on the air is also proportional to the mass of air moved multiplied by the pressure it is moved against, hence this pressure is proportional to the square of the velocity. Bearing these facts in mind, we can readily understand the more important considerations relating to the proper arrangement of air ducts.

It is evident in the first place, that the greater the sectional area of an air duct is, the lower the velocity at which a current at a certain number of cubic feet per minute will pass, and consequently the less work (in proportion to the square of the velocity) will be needed to move it. There is, however, little advantage in increasing the cross-section of a duct to much more than the cross-section of the fan opening, as additional velocity would then have to be given to the air as it passed through the fan, with corresponding increase in resistance. Nor would it, as a rule, be an advantage to increase the size of the fan, so as to permit of a low velocity through it, as in such a case the current is easily reversed by wind. The space occupied by the ducts and fan is also a material consideration. Any obstruction or narrowing in a duct will correspondingly increase the velocity, and therefore still more the resistance, so that all obstructions should be avoided, including those due to deposits of dust. Any sharp bend converts into heat the energy of motion possessed by the moving air, and an equivalent quantity of energy of motion has again to be communicated to the air beyond the bend. A sharp bend may thus double the resistance, and ought to be avoided if possible. A gradual bend causes much less extra resistance. A further factor in causing resistance is friction of the air along the sides of the duct. With the comparatively short and smooth ducts commonly used in factory ventilation, this factor is, however, small, and need scarcely be considered.

Roughly speaking, the cross-section of the duct or its combined branches should be about equal to that of the fan opening; and if a centrifugal fan, capable of overcoming considerable resistance, is used, the duct may be much smaller, and the air velocity in it much higher, than if a propeller fan is used. A propeller fan can only work against small resistance, while a centrifugal fan is adapted for much greater resistances. On the other hand, the horsepower required for the centrifugal fan is greater, in correspondence with the greater velocity of the air current and consequent greater resistance.

An important matter is the arrangement of branch ducts leading into the main duct connected with a fan. In exhausting dust-laden air from several dusty machines or dusty work-places, a corresponding number of branch ducts are required. If they are not properly arranged the amounts of air passing along the different branch ducts will differ considerably, so that at one place the exhaust current is too strong, and at another too weak. Unnecessary resistance may also be caused.

Dr. Haldane describes experiments made by the Home Office Committee to determine the best arrangement of branch ducts. The Committee found that 'the loss of power at the junctions, and at the same time the tendency to inequality in flow through the branch ducts, may be avoided by the simple expedient of making the branch ducts join at a slant.'

The concluding part of Dr. Haldane's paper deals with the relative merits of propeller and centrifugal fans for the removal of dust and fumes, and gave a number of illustrations of the application of the principles laid down to special cases. In concluding, he referred particularly to the method of increasing the effectiveness of exhaust openings by blowing air towards them. Though the system involves the provision of two fans, there are many cases, he things, in which the method could be successfully and economically used."—J. S. HALDANE.—*Engineering Magazine*, Aug. 1908, p. 757. (J. A. W.)

BACTERIA AS AGENT IN THE OXIDATION OF AMORPHOUS CARBON. — Lamp black, charcoal, coal and peat can all be oxidised by a diplococcus obtained from garden soil, and, no doubt, by other organisms as well. The action is only small, but it was quite clearly established (1) by passing a stream of air, freed from all traces of carbon dioxide over the material and determining the presence of carbon dioxide in the issuing gas by absorption with baryta and titration with oxalic and hydrochloric acids; (2) by determining the rise of temperature due to oxidation by means of a thermopile and galvanometer, and (3) by detecting, in the case of charcoal, the presence of calcium carbonate in the flasks inoculated with the bacteria. Oxidation only takes place under conditions favourable to bacterial activity; thus the rate increases with the temperature to 40°, but at 100° there is no oxidation at all. Similarly, oxidation ceases when the amount of moisture present is insufficient for bacteria. It follows that micro-organisms are entirely responsible, and the author refers to various consequences of such an action, suggesting, among other things, that the heat generated by the bacterial activity may be an important factor in determining the spontaneous combustion of coal, or firing a mixture of explosive gases in contact with it.—M. C. POTTER, Proc. Royal. Soc., 1908, 80, B. 239-259. —*Journal of the Chemical Society*, June, 1908, p. 524. (W. A. C.)

THE USES OF KIESELGUHR. — "According to *Engineering* the use of kieselguhr in Germany is now very extensive. The United States Consul at Chemnitz furnishes the following information concerning the deposits of infusorial earth in Germany and the articles manufactured therefrom:—Large amounts are employed in the manufacture of dynamite, where the remarkable absorbent properties of the material come into play. Its use alone as a fertiliser, and also in the preparation of artificial fertilisers, especially in the absorpion of liquid manures, is widespread. There is also an extended use of the earth for rapid filtration purposes, as well as for covering steam pipes, lining refrigerators, and filling the walls of fireproof safes. In the manufacture of water glass, of various cements, of glazing for tiles, of artificial stone, of ultramarine and various pigments, of aniline and alizarin colours, of paper, sealing wax, fireworks, gutta-percha objects, Swedish matches, solidified bromine, scouring powders, papier-maché, and a variety of other varieties, there is a large and steady growing demand. For some of the purposes in question, and especially when kieselguhr is used to absorb nitroglycerine in the preparation of dynamite, it is of prime importance that the earth should be freed as far as possible from moisture."—*Indian Engineering*, July 11, 1908, p. 27. (A. R.)

DURABILITY OF MODERN ELECTRICAL APPARATUS. —"Electrical apparatus, when constructed in accordance with the best engineering practice, is well-nigh indestructible if subjected only to the ordinary amount of wear and tear, and often comes out with flying colours from the devastating reign of floods and fires. The few incidents quoted below are excellent examples of the amount of hard usage that well-constructed electrical apparatus will survive, and srill be in working condition.

In the Knights Deep mine, near Johannesburg, South Africa, there were installed thirty-six 15 kw. transformers and twelve three-phase, 50 h.p., G. E. induction motors for operating the mine pumps. Through a peculiar combination of circumstances, the mine was allowed to fill with water—the motors and transformers remaining under several hundred feet of water for over two years.

When taken out, three of the motors were rewound in the local repair shop. The others were simply dried out and then soaked in oil. Contrary to all expectations and to the surprise of all concerned, the insulation was found to be in excellent condition, and the motors were put into service—apparently none the worse for their high-pressure bath. The transformers were treated in the same manner, and shortly afterwards they replaced several transformers of English make, which had burned out only a short time after being installed.

A railway power station situated near the Mississippi River experienced a severe flood during the season of high water. Two 1,500 kw. 6,000 volt generators were more than half under water for nearly a week, but when dried out carried their load without giving any trouble whatever. In fact, the first machine to be dried out carried the entire railway load until the second machine could be put into condition.

Another excellent example of the durability of the induction motor was shown by the remarkable performance of a G. E. 5 h.p. motor which was used in driving a pump in the G. H. Perry Co.'s quarry in South Dakota. The motor was often allowed to run without any attention over night, especially during the rainy season. One night a heavy rainstorm filled the quarry to such a height that the motor was half under water. When started the next morning, it picked up its load and carried it steadily until the pit was emptied of water. On examination it was found to be none the worse for its prolonged bath, although a new pulley had to be put on in place of the old one, which was softened and warped out of shape."—*Mines and Minerals*, July, 1908. (W.R.D.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

ROCKS AND ROCK MINERALS. A Manual of Elements of Petrology, without the use of the Microscope, for the Geologist, Engineer, Miner, Architect, etc., and for Instruction in Colleges and Schools. By LOUIS V. PIRSSON. Pp. iv. +414., price 10s. 6d. (New York: John Wiley & Sons. London: Chapman & Hall, 1908.)

"Many years' experience gained in teaching and in the field has enabled the author to furnish the student with a handy book of reference concerning the common rocks and rock-forming minerals and how they may be readily distinguished without the aid of the microscope or chemical analysis. Part I. is introductory. In Part II. the properties, description, and determination of rock-making minerals are dealt with; while Part III. is descriptive of the igneous, stratified, and metamorphic rocks, and also treats of their origin and determination. The book is illustrated by 36 plates of rocks and rock scenery, many of them excellent, but a few are cloudy and indistinct. In addition, there are 74 text illustrations of a somewhat meagre character.

While the author has incorporated considerable original material the work is necessarily mainly one of compilation. As affording a clear digest of the vast amount of important economical information containing in the bulky volumes of the Memoirs and Bulletins of the United States Survey, this handy and clearly printed text book is a welcome addition to petrological literature. Since the work is intended as much for the engineer, architect, and miner as for the general geological student, we miss several petrological terms of no direct practical value; but in the classification of the igneous rocks the author has not refrained from introducing the much discussed, but by no means generally accepted, terms of 'persalane,' 'salfemane,' 'dofemane,' 'perfemane.' With this exception, the book is not overburdened with useless technicalities, and if it contains some information of little practical significance, it also contains a great deal of indispensable knowledge, and much with which the engineer and miner should be familiar."—*London Mining Journal*, Aug. 29, 1908. (A. R.)

THE GEOLOGY OF COAL AND COAL-MINING. By WALCOT GIBSON, D.Sc., F.G.S., price, 7s. 6d. (London: Edwin Arnold.)

"This book is the first of a series of works on economic geology undertaken by the experienced geologist, and edited by Dr. E. J. Marr, of Cambridge. Such a series published in England would hardly fail to commence with the coal industry, by reason of its priority in magnitude and importance, and in this branch, as in some others, we could hardly have a more capable guide than Dr. Gibson. The subject in the volume before us is developed systematically. The author first deals with the constitution of coal considered as a rock, then passes on to its geological relationships, more especially in so far as their interpretation assists the work of discovery, and finally presents us with a description—of course a brief one—of the coalfields of Britain and of the world. A chapter on 'the study of a concealed coalfield' possesses a peculiar interest to geological students in a country particular rich in coalfields, the possible extensions of which constitute a factor of great economical importance. The singular consistency in the succession of the rocks of the carboniferous period and the rich development and marked individuality of their fossil flora, render them especially susceptible of independent treatment, and give a scientific completeness to Dr. Gibson's book, which, taken in conjunction with its other excellencies, makes it an admirable inaugural volume for the new series."—*The London Mining Journal*, Aug. 22, 1908, p. 250. (W. A. C.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 382/08. William Arthur Caldecott. Improvements appertaining to means for separating liquid from crushed ore products. 19.9.08.

(P.) 383/08. Bruce Edward Tennent. Improvements in the preparation of charcoal for the precipitation of gold from solutions. 19.9.08.

(C.) 384/08. Albert Donald Furse. Improvements in disintegrating apparatus. 21.9.08.

(P.) 385/08. Frederick Walter Baynes. Improvements in apparatus connected with pumping machinery and the like. 22.9.08.

(C.) 390/08. Charles Washington Merrill. Process of and apparatus for precipitating and recovering valuable materials from solution. 1.10.08.

(C.) 392/08. Charles Walter Eccleston. Improvements in ore concentrators. 1.10.08.

(C.) 393/08. Arthur Lumley. Improvements in washing apparatus. 2.10.08.

(C.) 394/08. Henri Poulain. Improvements in boring crowns and the process of manufacture of the same. 2.10.08.

(P.) 396/08. John Thomas Mitchell. A wire cutting machine. 3.10.08.

(P.) 397/08. Walter Edward Kimber. Improvements in operating the stamps of stamp mills. 7.10.08.

(P.) 398/08. James Frederick Ferguson. Improvements in furnaces for smelting gold slimes and the like. 9.10.08.

(P.) 399/08. Robert Copley (1), William Mahoney Butler (2). A mercury and amalgam trap. 9.10.08.

(C.) 400/08. Julian de Irabian. Turbine operated by air current. 10.10.08.

(C.) 401/08. Joseph Knight (1), Richard Steel Somers Bellamy (2). Improvements in or relating to air operated rock drills. 12.10.08.

(P.) 403/08. John Chisholm. A new invention with regard to insulating material. 13.10.08.

(P.) 404/08. Frederick Gwilliam. Improvements in apparatus for drilling or boring in the earth's strata. 15.10.08.

(P.) 405/08. Paul Montravel Tennick. Improvements in means for removing incrustation or scale from the interior of tubes, pipes and the like. 15.10.08.

(P.) 406/08. Charles Hansen. Improvements in safety brake arrangements for mine cages and other hoisting apparatus. 15.10.08.

(P.) 407/08. Kenneth Austin. Improvements in means for preventing winding accidents in mines and the like. 16.10.08.

(P.) 408/08. Alfred Stapleton Winton. Improvements in ore concentrators. 16.10.08.

(C.) 409/08. Lewis Condict Bayles. Improvements in percussive rock drilling machines. 16.10.08.

(C.) 410/08. Waldermar Hommel (1), Henry Livingstone Sulman (2). Improvements in the separation of zinc from its ores or compounds. 16.10.08.

(C.) 411/08. Albert Edward Hodder. Improvements in or relating to liquid-containers and valve mechanism, therefor particularly applicable for use in water and like elevators. 16.10.08.

(C.) 412/08. Francis Harrison (1), Christoph Heinrich Dorman (2). Process for producing tungsten metal combinations. 16.10.08.

(P.) 413/08. Antoine Henri Imbert. Improvements in and relating to electric furnaces. 16.10.08.

(P.) 414/08. Charles Graham Stone. Improved type of battery and apparatus for the reduction of ores. 17.10.08.

(C.) 415/08. Edward Lloyd Wynyard Bellhouse. Improved method of and means for securing the heads of bits used in percussive rock drills to their shanks. 17.10.08.

(P.) 416/08. Andrew James Macmunn. Improvements in obtaining nitrogen for the preparation of cyanogen compounds. 17.10.08.

(P.) 417/08. Robert Southerby Magson (1) Edwin Charles Edyvean (2). Improvements in renewable stamp guides. 17.10.08.

Changes of Addresses.

Andreoli, G., *l/o* Nigel ; P. O. Box 2012, Johannesburg.

Arderne, R. A. S., *to* 56, Carnarvon Road, Lorentzville, Johannesburg.

Beatty, J. W. S., *l/o* Luipaardsvlei ; Village Deep, Ltd., P. O. Box 1145, Johannesburg.

Blyth, W. B., *to* Great Fingall G. M., Ltd., Day Dawn, West Australia.

Brown, G. D., *l/o* Johannesburg ; French Bob's Mine, Louw's Creek, Barberton.

Butters, Chas., *to* 333, Kearney Street, San Francisco, Cal., U.S.A.

Chapman, F. T., *l/o* Springs ; Witwatersrand Deep, Ltd., P. O. Box 5, Knights.

Chew, B., *to* P. O. Box 6196, Johannesburg.

Cooke, R. C. H., *l/o* Rhodesia ; 17, St. Edmund's Terrace, Regents Park, London, N.W.

Crosse, Andrew F., *to* 33, Cambridge Square, Hyde Park, London, W.

Ford, Milverton, *l/o* Johannesburg ; Aurora West United G. M. Co., Ltd., P. O. Box 26, Maraisburg.

Harris, J. V., *l/o* West Nicholson ; Blanket Mine, via Gwanda, Rhodesia.

Hawthorne, J. W., *l/o* Germiston ; Kerang A., Victoria. Australia.

Hellmann, Fred, *l/o* London ; 530, Mills' Building, 15, Broad Street, New York, U.S.A.

Johnston, James, *l/o* London ; c/o Chas. Butters & Co., 333, Kearney Street, San Francisco, Cal., U.S.A.

Loevy, Dr. J., *to* 21, Luisenstrasse, N.W. 6, Berlin Germany.

Mackay, A. N., *l/o* London ; Cherokee Goldfields, Ltd., San Julian, via Parral, Chihuahua, Mexico.

Manners, C. E., *l/o* Nigel ; Angelo Mill, East Rand Proprietary Mines, Ltd., East Rand.

Milligan, J. T., *l/o* Klerksdorp : 91, Loveday Street, Johannesburg.

Mills, T. L., *l/o* Redruth ; Royal School of Mines, London, S.W.

Murphy, T. D., *l/o* Mexico ; Aguacate Mines, A San Mateo, Costa Rica, Central America.

Oliver, W. H., *l/o* Truro ; Broomassie Mines, Broomassie, via Tarkwa and Sekondi, West Africa.

Pearson, Thos., *l/o* Knights ; Cloverfield Mines, Ltd., P. O. Geduld Station.

Pope, J., *l/o* Gwelo ; New Goch Gold Mines, Ltd., P. O. Box 1096, Johannesburg.

Schwabe, S. B., *l/o* Johannesburg ; West Rand Consolidated Main Mines, Ltd., P. O. Box 38, Krugersdorp.

Sharp, W. P. F., *l/o* Johannesburg ; Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston.

Simpson, C. B., *l/o* England ; Geldenhuis Deep, Ltd., Cleveland.

Simpson, C. E., *l/o* Selukwe ; Concession Hill Mine, Hartley, Rhodesia.

Tom, Isidore, *l/o* London ; Associated Gold Mines of W. A., Kalgoorlie, W. Australia.

Urquhart, W. G., *l/o* Krugersdorp ; Simmer Deep, Ltd., P. O. Box 178, Germiston.

Ward, J. P., *to* New Goch Gold Mines, Ltd., P. O. Box 1096, Johannesburg.

Watson, E. S., *to* Simmer and Jack East, Ltd., P. O. Box 47, Germiston.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. IX. NOVEMBER, 1908. No. 5.

Proceedings
AT
Ordinary General Meeting, November 21, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday, November 21st, Mr. R. G. Bevington (President), in the chair. There were also present :—

38 Members: Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, K. L. Graham, C. B. Kingston, A. Richardson, C. B. Saner, Prof. G. H. Stanley, H. A. White, W. A. Caldecott, W. Cullen, F. Alexander, G. Andreoli, S. Beaton, E. Browning, F. W. Cindel, W. M. Coulter, J. P. Creed, L. Evans, J. C. Greer, Jr., F. G. Guthrie, J. Higham, C. B. Hilliard, G. A. Lawson, J. Lea, C. D. Leslie, G. Melvill, J. T. Milligan, S. Morison, S. S. Osborn, C. F. Parry, C. E. Rusden, A. L. Spoor, A. D. Viney, J. P. Ward, H. Warren, and J. Watson.

6 Associates: Messrs. J. R. Allan, J. Chilton, C. L. Dewar, D. Nicholas, Hy. Rusden, and C. B. Simpson.

10 Visitors and Fred. Rowland, Secretary.

The President: I beg to move that the minutes of the last ordinary monthly general meeting, as published in the October *Journal*, be confirmed.

This was seconded by Mr. C. B. Saner and agreed to.

NEW MEMBERS.

Messrs. F. F. Alexander and H. Warren were elected scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows :—

FURNER, EDWARD ARTHUR, Crown Reef G. M. Co., Ltd., P. O. Box 1145, Johannesburg. Mill Manager.
JOHNSON, ARTHUR JOHN, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Foreman Amalgamator.
JONES, WILLIAM HARRY RICHARDSON, Public Works Department, Standerton. Civil Engineer.
MARTIN, HENRY STUART, Messrs. H. Eckstein & Co., P. O. Box 149, Johannesburg. Consulting Engineer.
MINARDS, WILLIAM IVEY KENNEDY, Benallack, Grampound Road, Cornwall. Cyanider.
TONNESEN, OTTO STUHAUG, East Rand Proprietary Mines, Ltd., P. O. Box 65, East Rand. Mine Surveyor.
WILLIAMS, RICHARD ST. JOHN BARNARD, Hauser Tribute Co., P. O. Box 160, Johannesburg. Manager.

The Secretary announced that the following gentlemen had been admitted as Associates by the Council since the last general meeting :

GRIERSON, NORMAN VICTOR, Higginsville, Western Australia. Metallurgist and Mining Engineer.
JAMESON, CHARLES GODFREY, Jumpers Deep, Ltd., P. O. Box 1, Cleveland. Mining Engineer.

And as a Student :

BROWNE, CLAUDE MELVILLE, Cinderella Deep, Ltd., P. O. Box 75, Boksburg (Student at the Transvaal University College).

GENERAL BUSINESS.

Mr. E. H. Croghan (*Member of Council*): I should like to test the feeling of our Society on a certain matter, which, though it may not be scientific, would tend to produce a more sociable feeling at our meetings and indirectly benefit all of us, as well as the Society.

As a Society, good work is carried on, but it seems to be in the hands of only a few. This is a very bad state of affairs, and to my mind is primarily due to a lack of intercourse not only amongst our members individually but also between the members and those controlling the Society's affairs. A society should be like a club where everyone is on an equal footing, and no one should feel that by virtue of his worldly position he is debarred from discussing a subject, no matter if trivial, with those with whom he desires to. Under our system of ordinary monthly meetings, it is difficult for it to be otherwise, in view of the short time at our disposal. We do have as many excursions as possible, but I wish to go some steps further. Sport has done much in cementing the British Empire, well, why should it not be entertained in connection with science, locally at any rate. It would be indirectly a means of interchanging ideas in a more casual way, some of which could be as usual contributed officially to the Society. I feel assured many sleeping geniuses would then better develop, and come a little more into daylight, thus promoting our mutual interests. The idea could be extended still further by inducing other technical societies to enter the arena: we might also arrange smoking concerts, as I am sure we are endowed with plenty of talent. Certain annual functions might be held by the combined technical societies, we could have, say, a combined banquet or dinner, ball or dance, and athletic sports, trophies for which, I feel sure, would be given by our mine owners and others.

In conclusion, though this is an extremely unusual subject to mention at such a meeting, I think you will pardon me, as it is the only definite opportunity one has of bringing such matters forward.

The President: This is certainly a new idea of Mr. Croghan's for providing for social intercourse between the members of the Society, and as he has launched the bolt, I think it is up to any member of the Society to follow it up for further consideration or otherwise. Would anyone like to make any remarks?

PROPOSED LOCAL SECTIONS.

The President: There is, at the end of the agenda paper, a notice of a Special General Meeting to be called for the purpose of considering some proposed additions to the constitution and rules. It may not be known to many of you why these additions are wanted, but I may tell you that our members are so widely scattered in various parts of South Africa, and other countries, that suggestions have been made with regard to the formation of local centres or sections. We have had correspondence from some of our members in Rhodesia who wish to be able to form a local centre there, in order that they may meet one another and discuss matters connected with their mining operations. Some members also have an idea of forming a branch in another country. Therefore, your Council considered that in order to place it in a position to deal with the formation of local sections, your approval should be asked of the proposed additions to the constitution and rules. This matter will come up at a Special General Meeting on the 19th of December, at 7.30 p.m., and I trust that all members will take the matter into consideration and will be present and vote either for or against, so that we may know the feeling of the Society. Suppose the additions are made to the constitution and rules it will not, at any rate, hurt anyone. We may never have to avail ourselves of these additions, but in order to prepare for such a case—and I think we should be prepared for it, as a feeling does exist that local sections may be formed—we propose, with your sanction, to make these additions to our constitution and rules.

THE SILVER COATING OF AMALGAMATING PLATES.

Mr. W. A. Caldecott (*Past-President*): Whilst electro-plated copper plates have never come into general use on these fields, and whilst some of the virtues attributed to them may have little better foundation than the fond fancies of the old-time millman, there is yet no doubt that the percentage recovery by amalgamation in the early stages of starting a new mill with plain copper plates is by no means satisfactory. When more stamps or tube mills and shaking tables are installed on a plant already in operation, a little amalgam can usually be spared from existing plates to set the new plates, but even so, greenish stains and patches for some time offend the amalgamator's eye and defer the day when he can view with satisfaction a uniform silvery surface persisting from one dressing till the next.

The use of silver amalgam for setting purposes has frequently been recommended, and I have experimented upon it in various ways, dating from early but unsatisfactory attempts, which involved much labour in filing silver coins into powder. The simple method finally adopted was based upon obtaining a pure silver amalgam of buttery consistency containing the silver in the finest possible state of division. If such an amalgam is applied to plain copper plates after the usual scouring some two or three weeks before the plate is put into commission and the plate frequently dressed with the same amalgam during this period, the silver amalgam is given the opportunity to become thoroughly incorporated with the surface of the plate, with consequent benefit to amalgamation when the stamps or tube mills are started.

Silver amalgam is now commercially obtainable, but if preferred, it can be prepared in the manner described by Louis, as follows*:—

"A sufficient quantity of silver coin (about $\frac{1}{4}$ oz. per sq. ft. of surface of the tables) is dissolved in dilute nitric acid in a porcelain basin with the aid of a gentle heat. The solution is evaporated to dryness very gently, preferably over a water bath, and then heated till the saline mass commences to fuse, and till all its bluish tinge is turned to greyish-black, this change indicating that all the soluble cupric nitrate is decomposed, insoluble cupric oxide being left behind. The salt is then dissolved in a small quantity of water and filtered into a jar or beaker. Pure mercury, to the weight of about three times that of the silver used, is poured in, a few drops of nitric acid added, and a few pieces of bright iron floated on the surface of the mercury. The silver will at once commence to precipitate and be absorbed by the mercury forming silver amalgam, the process taking a few days to complete thoroughly. The silver amalgam so produced should be of a pasty consistency."

---

* "Handbook of Gold Milling," 1899, p. 313.

In carrying out the foregoing method, it will be found that small silver coins dissolve more readily than the larger ones, and that small wire nails provide the iron required in a convenient form. In the process an iron-mercury couple is formed, which serves to deposit the silver present in solution upon the surface of the mercury which immediately absorbs it, whilst a corresponding amount of iron is dissolved.

If pure silver can be obtained, the evaporation of its solution in nitric acid, with subsequent heating to remove the copper, may be omitted and the process thus simplified, but if silver coin containing copper is used as a ready though somewhat expensive source of silver, and the copper not removed, the resulting silver-copper amalgam will show a greenish surface film soon after application to the plates, owing to oxidation of the copper present. The molecular state of fineness of the silver in the amalgam permits a much more perfect coalescence between the surface of the copper plate and the amalgam than when coarse particles of metallic silver are present in the latter. To ensure, however, a silver amalgam coating which will not tarnish on exposure to the air, time for absorption must be allowed, and for this reason the preparation of the plates some time before use is recommended. After milling has started the coating of silver amalgam is gradually removed from the plates as gold amalgam is scraped off, but by the time the former is all gone its purpose will have been served and its place taken by an equally efficient and permanent coating of gold amalgam.

The cost of silver coating copper plates in the manner described is relatively small as compared with the value of the amalgamable gold which otherwise passes away with the tailings in the early stages of crushing, and the allowance of $\frac{1}{4}$ oz. of silver per square foot of plate area is certainly liberal, as in my own experience a considerably less proportion than this has sufficed.

The following micro-photographs of ordinary copper plates, coated with silver amalgam, were kindly prepared for me by Mr. McA. Johnston.

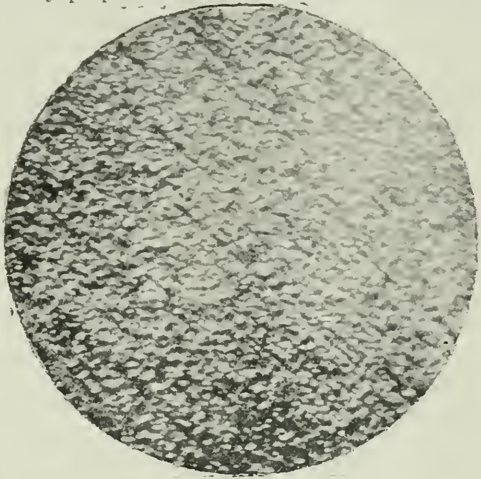


FIG. A.

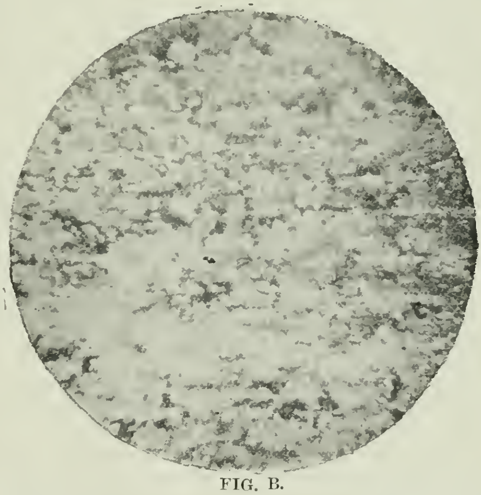


FIG. B.

Fig. (A) is a magnification of 36 diameters of the surface of the copper plate after having had silver amalgam applied to it once. The white portions are the globules of mercury standing out in relief, whilst between many of them the copper surface of the plate is visible quite untouched.

Fig. (B) is a magnification of 36 diameters of the surface of the copper plate after having had silver amalgam applied to it three times, the whole treatment consisting of four burnishings. Note here the more extended surface of the amalgam whilst in no part is the copper visible.

[Samples of the copper plates described in this note together with micro-photographs were passed round the room.]

The President: In amplification of the author's remarks I may say that one reason for preparing a plate with silver amalgam is, amongst other things, a matter of cheapness, as its cost is comparatively small. A copper plate takes silver amalgam very readily, and once it is coated it does not tarnish, the result being that as soon as the battery is started the plates are in condition to catch gold at once. Of course, in scraping, a certain amount of silver will come off with the first gold amalgam, but that does not matter.

THE GASES RESULTING FROM THE USE OF HIGH EXPLOSIVES.

By Wm. Cullen (Past-President).

Some few years ago I ventured to read a paper on Miners' Phthisis—a subject of which I knew very little and on which I had practically no first hand information. The adage "that fools rush in where angels fear to tread" was at that time particularly applicable to me, but I am far from regretting the writing of that paper, as it was the means of stimulating enquiry in this particular subject, and I think I may safely assert that the direct result of the enquiry has been an all round improvement in the working conditions of those who labour in the bowels of the earth. Drs. Moir, Macaulay, and Irvine and Mr. Heymann, deserve our unstinted thanks for all they have done in our immediate sphere, and outside of it, the work of Dr. Haldane and Mr. Mann, of West Australia, has not been without influence. No one of these gentlemen would venture to assert that anything like finality has been reached, and I propose to-night to add another chapter to our knowledge. I shall have to go back on opinions which I have expressed on more than one occasion, here and elsewhere, and I am prepared to be told that both my facts and my deductions are wrong, but I shall deal principally with the former. I cannot say that I found these facts particularly palatable, and I do not expect that my mine manager friends as a body will like them either, but it is because they are facts that I think they should be made known as widely as possible. In the light of the figures which I am going to place before you presently I am more convinced than ever that the "ventilation of our mines" is the most important problem which we have to face at the moment, and this not only from a hygienic but also from the economic point of view, because the one is inextricably bound up with the other. A friend of mine—a mine manager of no mean repute—told me the other day that he came out to this country 10 years ago with 18 miners, to work on a certain mine. To-day only he is alive. All the others are dead.

This paper is one of a series of three, and it gets first place simply because I considered it the most urgent. It will be followed at a convenient interval by another by my colleague, Dr. Weiskopf, on the methods of analysis which have been employed in arriving at the figures which I give in this paper, and the third paper, a joint one, will treat of a new method of using explosives. I may, however, have to supplement this paper by another if I have an opportunity of carrying the investigations further.

Before proceeding I should like to refer very briefly to the results obtained by investigators in the same field of enquiry. Dr. Moir and Mr. Heymann working for the Phthisis Commission found that certain samples of mine gas contained an amount of carbon monoxide which was not only dangerous, but deadly. I was inclined to doubt their figures at the time, and received a certain support for my contention from the investigations of Mr. Mann, which followed shortly afterwards. Mr. Mann found no trace of carbon monoxide in the many experiments which he carried out, but found indications of nitric peroxide—by the starch iodine test—in several instances. This, to say the least of it, was peculiar. The work of Dr. Haldane has no particular bearing on our own conditions, but his results seemed to confirm those obtained by Mr. Mann. Reverting, however, to the early work of our local investigators, it is only right to say that at that particular time the quality of explosives used on these fields was, comparatively speaking, bad. The local product which was mostly used had been lying in magazines during the war period and the imported article was in the same plight. Old explosives do not give good results whichever way one looks at it. Since that time, quality has improved all round, and I question if any mining field in the world is better supplied than the Rand is at the present moment.

The explosive mostly used on these fields is blasting gelatine; practically no other is employed in driving, developing and shaft sinking; my work has been mainly confined to the gases resulting from the use of this explosive, occasionally slightly modified on lines which will be indicated. On being completely detonated, only carbonic acid, vapour of water, and nitrogen are formed—all comparatively innocuous gases. The particular equation which shows this result has been before you frequently, and I may say here that it is confirmed by explosion inside a steel bomb. Incidentally, I may mention that the cartridge wrapper, which Dr. Moir assumed took part in the explosion and led to the production of carbon monoxide, does not do so, at least not to any extent, as I have demonstrated more than once. This, however, is a side issue. My contention has always been that if the conditions of the steel bomb were imitated as nearly as possible in practice, we should get results which at least approximate to the theoretical, *i.e.*, if the explosive is carefully tamped and certain other precautions taken. In no case was I able to get results which have the slightest resemblance to the theoretical, and with these introductory remarks I shall now proceed to describe the conditions under which the shots were fired, and the samples of gas taken. The experiments were carried out on more than one mine, and to certain unnamed mine managers I would express my best thanks for all the facilities which they gave me.

All the experiments were conducted in drives, *i.e.*, on development work, and a sample of the mine gas was generally taken before blasting—at a certain point. Immediately after the blast and before any air was turned on, the sampler proceeded to the same spot, generally 30 ft. to 40 ft. from the face, wearing a safety helmet, of course, and again sampled the gases from the blast. The actual sampling was done as follows. At the point fixed on, the sample bottle was opened, air was introduced into it by means of the bellows, which were emptied and filled 50 times. The same operation was gone through after the explosion. Scores of samples of mine air have been analysed, and although in most cases I have been given permission to make use of the figures, I do not propose to avail myself of this privilege to any extent. I may say, however, in passing that most of the samples were very bad indeed, though it is only fair to add, that in many cases those samples were taken from particular parts of the mine which were known to be bad. The astonishing thing about all these analyses is the almost universal presence of carbon monoxide, and that, more often than not, in dangerous quantity. The few examples which I give will however speak for themselves.

Before dealing with the experiments, I think it advisable to explain the underlying ideas. In the first place, there is a general impression that old explosives give much worse results than new; and that is so, as most people know, when we speak of the age of explosives in terms of years. Then again, there is a difference of opinion with regard to the best size of detonators—some contending that better results are obtained with No. 8 than with No. 6 and *vice versa*. With regard to tamping I hold rather extreme views. I have always had the idea that the current practice of employing sand cartridges was wrong in principle on account of the friable nature of the sand. Clay, to my mind, was infinitely preferable, so a comparison was tried. Incidentally, I may mention that Mr. Leslie, of the Simmer and Jack, who holds similar views to my own employs a variety of clay, and asserts that he gets much better work in practice. Then there was the vexed question of the benefit, or otherwise, of the so-called neutralisers, or antifume mixtures, of which there are sundry varieties on the market. There were various minor points, but the most important of all was to ascertain whether better all round results could not be obtained from a blasting gelatine which was so modified as to contain within itself more than the normal amount of oxygen. These experiments are unfortunately not yet concluded, but they have already given some most valuable data.

Now regarding all these points, I look upon the analyses of the resultant gases as the most accurate means of measuring the work done. In other words, the more completely the explosive is resolved into the simple gaseous mixture of vapour of water, carbonic acid and nitrogen, the better should be the footage, to use a technical term. Certain experiments seem to incline this way, but the results are not conclusive, and the experiments must be continued. What is conclusive is, that no matter how perfect the conditions were, very large amounts of carbon monoxide were always formed, and incidentally, I may say, that the conditions under which the experiments were carried out were above the average, in so far as exercise of care is concerned. When good results were sought, the best explosive obtainable was employed, the holes were all carefully charged and rammed home, tamping was done as well as it could be, detonators were known to be good, and all round the conditions were, as I have already said, above the average. Now for a few of the results.

In every case when not otherwise specified the tamping used was the ordinary sand cartridge.

TABLE I.—RESULTS OBTAINED FROM BLASTING ROUNDS.

No. of Experiment.	Explosive.	Detonator.	Distance from face.	GAS COMPOSITION.		Ratio of CO to CO_2.
				% CO.	% CO_2.	
1*a*	Ord. blast. gel.	6	35′	·012*	·134*	
1*b*	3 months old			·88	6·92	1—7·8
3*a*	Newly made	6	27′	·026*	·328*	
3*b*	blasting gelatine			1·09	6·56	1—6
6*b*	Ord. blast. gel. 3 months old	8	44′	1·28	8·07	1—6·3
4*a*	Blast. gel. 3 mon.	6	35′	·021*	·299*	
4*b*	old w. neutraliser			·90	7·44	1—8·2
2*a*	Special composi-	6	40′	·013*	·50*	
2*b*	tion, blast. gel.			·56	3·7	1—6·6
5*a*	Special composi-	6	40′	·011*	·216*	
5*b*	tion, blast. gel.			·73	5·81	1—7·9
12*a*	Modification of 2	8	16′	·007*	·123*	
12*b*	,,			·718	6·05	1—8·4
13*a*	Modification of 5	8	20′	·026*	·058*	
13*b*	,,			·66	6·20	1—9·3
14*a*	Different make	6	36′	·035*	·25*	
14*b*	of blast. gelatine			·467	4·00	1—8·5

a and asterisks indicates before blasting. *b* after blasting.

Table I. deals with a series of experiments which are almost self explanatory, but in calculating the ratio of CO to CO_2, the figures of which appear in the last column, I have made no correction for the amounts present in the gases before the blast, though in some of the cases it is evident that they would materially affect the ratio. My reason for not doing so is that I do not think there is sufficient data to go upon, and one can play with figures too much. The one point which is established beyond the shadow of a doubt, is, that carbon monoxide is always formed in large, even dangerous quantity. The presence of the large percentage of carbonic acid calls for no remark, and where larger quantities than the usual are indicated, the natural conclusion is that the sample has been obtained in a concentrated form. Strange to say, no trace of nitric peroxide was found in any of the samples, though we would not be justified in asserting that it was not formed. I think myself that it must have been formed in every case.

To come now to the actual results: 1, 3 and 6 are rather an interesting trio to compare. I fully expected that both 3 and 6 would show up better than 1, but they do not. Freshly made explosive is certainly more sensitive than explosive which has been stored for some time, and the Trautzl tests also indicate greater strength, but the results do not bring this out. The natural explanation with regard to sensitiveness is that the No. 6 detonators employed here are sufficiently strong to counteract any tendency towards insensitiveness, within a storage age of three months. Again, the prevalent idea is that No. 8 detonators give better results than No. 6, and although the experiments do not show this (Experiment 6), I think the practice of using No. 8 is sound, as it increases the safety factor for practical work. No. 4 is an experiment which stands by itself. The neutraliser was employed according to the inventor's instructions, but it cannot be said that the results are any improvement over ordinary practice. Incidentally, I may say that I never could see on scientific grounds why any of these so-called antifume mixtures or neutralisers could have any effect one way or another beyond perhaps diluting the gases with a certain small quantity of oxygen.

Experiments 2, 5, 12 and 13 were carried out with an explosive which is the subject of certain patents, and for which certain advantages are claimed. The results, with the exception of 13, do not indicate any improvement, but they are not complete. The percentage of the special oxygen yielding ingredients in Nos. 2, 5 and 12 are very small. In 13 they are increased, and by still further increasing them I am hopeful that it will be possible to produce an explosive which will be more suitable for the work on these fields. I may say that all of them, but especially 13, have given excellent results in practice, *i.e.*, as measured by footage, and that on many mines. No. 14 is a make of blasting gelatine produced outside the Transvaal, but the results given call for no special remark.

Taken as a whole, therefore, the main conclusion which one is justified in making from these figures is :—(*a*) That carbon monoxide is always produced. (*b*) That the blasting gelatine employed in 13 indicates a slight improvement over the others.

Appendix I. gives the actual working conditions in greater detail than is shown in the table (No. I.).

On the question of tamping most divergent opinions are held. Several authors have referred to this question, among them being Mr. Weston at our last meeting. I do not propose to review these opinions here, and content myself by again expressing the personal view that sand is not a suitable tamping material. I believe that good clay is infinitely better, and if trouble were taken to procure it and to make it up into proper cartridges much better results would be obtained in whatever way one looks at it. The experiments given in Table II. are only of a preliminary nature, but they are interesting as indicating a decided improvement in favour of dagga—which as you know is at least an approximation to clay. The tamping cartridges were made up in the usual way with the dagga dry, but they were used damp, which makes it somewhat sticky or adhesive. The details are given in Appendix II.

TABLE II.—INFLUENCE OF TAMPING.

No. of Experiment.	Tamping.	Distance from face.	Gas Composition.		Ratio of CO to CO_2.
			% CO_2.	% CO.	
15*a*	Dagga cartridges	47′	·049*	·356*	1–7·93
15*b*			·58	4 60	
16*a*	Ordinary sand cartridges	47′	·050*	·258*	1—6·5
16*b*			·28	1·84	

a indicates before blasting.
b indicates after blasting.
Ordinary blasting gelatine used with No. 8 detonators.

In a later paper I propose to deal with the subject of blasting by electricity, and the figures given in the next Table III. have only reference to a phase of that question. We all know that comparatively large amounts of carbon monoxide are produced from the burning of fuse, and if the details of Appendices I. and II. are examined it will be seen that the amount used for the blasting of a round is from 72 ft. to 96 ft. I shall deal with this presently. The cut is generally drilled so as to make the holes converge at the bottom, and this in practice means that the cut comes

TABLE III.—RESULTS OBTAINED FROM BLASTING CUTS.

No. of Experiment.	Explosive.	Detonator.	Distance from face.	Gas Composition.		Ratio of CO to CO_2.
				% CO.	% CO_2.	
8*a*	Blasting gelatine	8	12′	·014*	·35*	
8*b*	3 weeks old			·195	1·63	1—8
9*b*	Blasting gelatine 3 weeks old	elec dets	27′	·208	2 03	1—9·7
10*a*	Blasting gelatine	elec	27′	·017*	·23*	
10*b*	3 months old	dets		·205	1·72	1—8·4
11*b*	Blasting gelatine 3 months old	8	27′	·22	1·71	1—7·7

a and asterisks indicates before blasting.
b indicates after blasting.

away at one blast, or at least so nearly so that one cannot detect any interval between the shots. I had an idea that firing by electricity, which ensured the simultaneous detonation of all the shots, would give better results, and the figures of this table, notwithstanding the fact that the electric detonators were only No. 6 as against No. 8 for the ordinary, certainly indicate a little improvement. Against this is, of course, the production of carbon monoxide from the burning of the fuse, but from other experiments it does not appear that the amount from this source could seriously vitiate the results given in the table. You will observe that in No. 10 only three holes were fired as against four in all the others. The details of these experiments are given in Appendix III.

The results of the foregoing experiments were so surprising and unlooked for, that I next decided to try gelignite, which is used for development on some mines. Gelignite may be described as a low grade blasting gelatine to which is added organic matter in the form of wood pulp, etc., and nitrate, so proportioned that the oxygen from the nitrate is capable of oxidising all the organic matter. Whether this oxidation is complete or incomplete has never been determined in practice, but judging from the analogy of ordinary black powder one would be safe in assuming that the combustion would not be complete. I shall not give the details in this instance, and shall content myself by stating that everything was done on exactly the same lines as in the former experiment, No. 6 detonators being used and the ordinary length of fuse.

For the results obtained, see Table IIIa. in next column.

So far I have dealt only with percentages of CO and CO_2 in the resultant gases, though most of the samples were completely analysed. These analyses enable us to calculate the composition of the gases produced in the borehole. A few of these results are given in the following Table No. IV.

TABLE IIIa.

GELIGNITE.	Before blast.	After blast.
Oxygen	20 310	17·96
Nitrogen	79·300	74·52
Hydrogen	nil.	·11
Carbon Monoxide ...	·028	1·23
Carbon dioxide ...	·362	6 18
Olefines	nil.	nil.
Oxide of nitrogen ...	nil.	nil.
	100·000	100·00

Proportion of CO to CO_2
1 to 4·9.

This is an astounding result.

TABLE IV.—CALCULATED COMPOSITION OF GASES PRODUCED IN THE BORE HOLE.

Gas.	12*b*.	13*b*.	14*b*.		Gelignite.		Ordinary Tamping.	Daaga Tamping.
Carbon monoxide	7·64	5·43	5·81	7·49	10·39	10·99	8·55	7·2
Carbon dioxide	63·52	57·16	50·42	65·03	50·30	55·81	63·94	64·5
Nitrogen ...	27·15	36·34	43·49	33·16	37·98	35·58	26·61	26·4
Oxygen ...	nil	nil	nil		nil		nil	nil
Hydrogen ...	1·50	·98	nil		·95	1·01	1·40	2·0
Total ...	99·81	99·91	99·72	105·68	99·62	103·46	100·50	100 1

It will be observed that under the heading of "14*b*" and "gelignite" two columns of figures are given which represent two methods of calculating the result. The first column represents what I might call the "oxygen" basis and the second "carbon oxide" basis. Those acquainted with gas analysis calculations will understand what I mean, so I need not enlarge on the point, It is evident, however, that the first, or "oxygen" basis method, appears to be the more accurate, but the difference between them would not affect the general conclusions in any way. I should also explain that the headings of the different columns in this table represent the same explosives as those referred to in Tables I. and II. The figures given are also the same, but differently stated.

According to theory, 12*b* and 13*b* should have furnished free oxygen—particularly 13*b*, but they do not. This is hardly surprising in view of the fact that the results obtained all through deviate so much from the theoretical. I shall return to this point later, as it is of great importance, but meantime I would again direct your attention to the high CO/CO_2 ratio of 13*b*—the highest so far found. It is stated in another way in Table I, but does not show up quite so high there, though the difference is not great. Unfortunately, the gases from experiment 4*b*, which, as you will recollect, was that one with which the neutraliser was used, were not completely analysed, but we have sufficient data to enable us to calculate the approximate composition of the borehole gases—which show no free oxygen. At this stage I may say that the theoretical gases given off by a scientifically constructed blasting gelatine are as follows :—

	With water as vapour 25· 635 mill. % vol.	No water condensed. % vol.	All water condensed. % vol.
CO_2 ...	69·84	61·7	72·52
N ...	26·46	20·8	27·48
H_2O ...	3·70	18·0	nil.
	100·00	100·0	100·00

I now pass to another aspect of the question, viz., the effect of burning fuse on the composition of the mine gases. This ground has already been pretty well covered by Dr. Moir, so I shall confine myself to a record of one or two rather interesting experiments.

In most of the experiments in Table I. about 96 ft., or four coils, were employed for blasting the round. This amount of fuse was burned in a drive, and the gases were sampled before and after, just as in the other experiments. Unfortunately the "before" sample met with a misadventure, so I can only give the results of the "after" which were

CO ... ·050%
CO_2 ... ·336%

As they stand these figures are not worth very much, excepting in so far as they show that the CO produced by the fuse is more than insignificant in amount,— indeed, this amount is highly dangerous. Of course, all the other results already given, must be looked upon as the combined effect of a fairly constant quantity of fuse with a fairly constant quantity of explosive. Even in the case of the "cut" blasts, Table III., the fuse was reduced pretty much in the same ratio as the explosive, taking the figures given in Appendix I. as the basis of comparison.

The records of some interesting experiments are given in Appendix IV. The figures will, of course, vary with the quality of the fuse, and other factors will have an influence, but assuming that the toxic effect of a certain percentage of carbon monoxide on human beings is known, it is easy to reckon by the rule of three how many persons one could kill by a certain number of coils.

The examination of every-day mine gases with regard to their vitiation is outside the scope of this paper, and no useful purpose will be served by directing attention to particularly bad samples. Although, as already said, I have been given full permission to make whatever use I like of certain figures, I do not propose to avail myself of this privilege, for the simple reason that as far as possible I desire to avoid controversy. This much, however, I can say in general terms, and in saying it I am also expressing the opinion of the managers concerned, that some of the places were unfit for human beings to work in, and that it was wonderful how they managed to do so. On the other hand, some of the analyses indicated a purity almost equal to the atmosphere at Sea Point, or the Durban Bluff, but while gladly admitting this, I wish to make it very clear indeed that I think the subject of ventilation is not receiving that amount of consideration which it ought to do in the light of the advances which are being made in other directions. For instance, the Sea Point atmosphere was generally found where a less pretentious one would have served the purpose equally well. I believe that few of those responsible for the practical operation of our great mining industry realise the great importance of the subject, and what economic advantages improved ventilation would bring in its train. But how could they realise, one might well ask? The subject has been surrounded by controversy, results have been questioned, the samples were unfairly taken, the author showed bias, and so on, but the most important of all is that the facts which I have placed before you to-night were not known—or put it in another way if you please—were not realised. They introduce new factors which are unpalatable to me, and equally so to many of my friends and colleagues on these fields. I therefore appeal to all those who are interested, to start the consideration of this subject afresh, to look at it from the broad as well as from the individualistic standpoint, and to show a genuine anxiety to get at facts. I know an appeal such as this is unusual in a scientific paper, but the subject is so very important, that I have ventured to do an unusual thing.

I now turn to one or two considerations which arise out of the figures which I have placed before you. You will recollect my reference to Mr. Mann's experiments—mostly with gelignite by the way—which showed an entire absence of CO. I believe, in the light of my results, that his methods of estimating it were at fault, and I am afraid that the same may be said of some of the work which has been done here. Whether or not, I have had doubts raised in my mind on many important points, which have hitherto been looked upon as settled. For instance, are our compressor gases as free of CO as we suppose, and have not mine gases frequently been reported as free of CO when they actually contained it? Remember that we are dealing with infinitesimal quantities of CO, that few people in this country know much about gas analysis, and that fewer still have the necessary apparatus for analysis. It is for this reason that you will have the full details regarding my figures placed before you at the earliest possible date. In no case was nitric peroxide observed, even in traces, though I believe, as I have said already, that very small quantities may have been produced. It is certainly produced when burning takes place, but all the experiments were carried out with great care in order to eliminate this complicating element. As a matter of fact, there was no reason for supposing that burning tock place in any one of the experiments.

Then again, I think the whole question of ventilation of our mines has been complicated by the figures which I have placed before you to-night. Undoubtedly, it makes the problem more difficult of solution, but the first stage is to get at the facts and to ascertain what is the true average state of the air at every part of the mine. This will not be done in a day or in a month, and it will cost a lot of money, but ultimately, I am sure those responsible will feel as interested in the analysis of their mine gases as they are in the "grade," or the monthly tonnage, for improved ventilation must ultimately mean reduced costs.

It has been represented to me that all the gases were sampled under very abnormal conditions, and that in order to make my case complete I should have sampled the drive again at certain time intervals in order to show the progressive decrease of CO. It must, however, be apparent that this would have been equally abnormal, so I think the only fair thing to do is to sample under working conditions before the blast and then after. They give one the two extremes, but the really important one after all is the one "before."

The conclusions at which I have arrived have been referred to during the course of this paper, but they may again be summarised as follows:—

(1) Carbon monoxide is produced in large quantities on the detonation of blasting gelatine, under all the ordinary conditions of local practice.

(2) The so-called neutralisers or antifume mixtures do not appear to have any influence one way or another.

(3) Clay tamping seems to give better results than ordinary sand tamping.

(4) Freshly made blasting gelatine gives no better results than that which is three months old.

(5) Within an age limit of three months a No. 6 detonator seems to answer as well as a No. 8.

(6) Blasting gelatine of the nature of 13*b* seems to give better results than the ordinary blasting gelatine.

(7) Firing by electricity seems to give slightly better results than ordinary time fuse.

(8) The ordinary quantity of fuse used vitiates the mine atmosphere to a very large extent.

No one is more conscious than I, that only the fringe of this subject has been touched, and further and equally interesting experiments are now in progress. If they are not completed when my reply is due, they will be made the subject of another communication.

With regard to No. 6, I have accumulated a large amount of data, all going to show that a greater footage is obtained with an explosive of this class, but I am diffident about placing it before you, as I know how liable we all are to be favourably predisposed, and if a miner thinks he is going to get better results than usual he can generally get them. The concensus of opinion with regard to the gases, however, is that they are very much better than my figures would indicate, and I am very hopeful that as a result of these experiments, a class of explosives will shortly be put on the market which will give at least as good footage as ordinary blasting gelatine, but very much better gases.

With regard to tamping (No. 3), it is difficult to say what amount of improvement the figures which I give will mean in actual practice, but it is sufficient for my purpose to demonstrate an improvement in the gases.

Regarding the results as a whole, it is very pertinent to enquire why they should deviate so much from the theoretical. That is a difficult question to answer, but the following considerations are worth bearing in mind. When nitroglycerine, say, is ignited in a steel bomb of convenient size, the resultant gases are in accordance with theory. If my recollection serves me aright —it is many years since I made the experiments—blasting gelatine behaves similarly. For my purpose the first case however is sufficient. Would nitro-glycerine in the form of dynamite give the theoretical results in our ordinary Rand practice? I do not think so (but intend to try), because the conditions are so entirely different. Then again, I have always had an idea that the varying burdens in the different shots of a round might account for the production of varying amounts of so-called noxious gases, but from the results of Table III. it is evident that this cannot be so. In the case of the electric detonators all shots were set off at the same time, and yet the amount of CO produced is relatively the same as in a round. No, it appears that we must look elsewhere for the possible causes, and these are now being investigated.

In conclusion, I should like to express my personal thanks to Mr. Waters, technical representative of my company, who has done practically all the underground work, and to my colleagues, Dr. Weiskopf and Mr. Greig, who have carried out all the analyses and done nearly all the calculations.

APPENDIX I.

Experiment No. 1.—Total depth of holes, 83 ft.; total No. $1\frac{1}{4}$ in. plugs, 79 = 43·8 lb.; temperature of level, 24·5° C.

Experiment No. 2.—Total depth of holes, $89\frac{1}{2}$ ft.; total No. $1\frac{1}{4}$ in. plugs, 78 = 43·3 lb.; temperature of level, 23° C.

Experiment No. 3.—Total depth of holes, $89\frac{1}{2}$ ft.; total No. $1\frac{1}{4}$ in. plugs, 82 = 45·5 lb.; temperature of level, 22·5° C.

Experiment No. 4.—Total depth of holes, $88\frac{1}{2}$ ft.; total No. $1\frac{1}{4}$ in. plugs, 76 = 42·2 lb.; temperature of level, 22·5° C.

Experiment No. 5.—Total depth of holes, $90\frac{1}{2}$ ft.; total No. $1\frac{1}{4}$ in. plugs, 78 = 43·3 lb.; temperature of level, 22·5° C.

Experiment No. 6.—Total depth of holes, 92 ft.; total No. $1\frac{1}{4}$ in. plugs, 84 = 46·6 lb.; temperature of level, 22° C.

Fuse used for all the foregoing, 96 ft.

Experiment No. 12.—Total depth of holes, 76 ft.; total No., $1\frac{1}{4}$ in. plugs, 81 = 45 lb.

Experiment No. 13.—Total depth of holes, 72 ft.; total No. $1\frac{1}{4}$ in. plugs, 77 = 42·8 lb.

Experiment No. 14.—Total depth of holes, 76 ft.; total No. $1\frac{1}{4}$ in. plugs, 78 = 43·3 lb.

Fuse used in each of the foregoing three cases, 72 ft.

The vertical depth in each case was between 1,400 and 1,700 ft., and the levels were all about 8 ft. × 5 ft. The tamping in all cases consisted of three 14 in. sand cartridges.

APPENDIX II.

Vertical depth to level	1,900 ft.
Distance from shaft to face ...	600 ft.
Size of level	7 ft. × 6 ft.

Dagga Tamping.—Three cut holes were charged with 24 plugs blasting gelatine and fired. The air was then turned on, and after about half an hour a return was made to the face and the air was sampled previous to blasting the round—which consisted of ten holes, averaging $5\frac{1}{2}$ ft. in depth. The holes were charged with 66 plugs, $1\frac{1}{4}$ in blasting gelatine = 36·6 lb.

Temperature of level	25° C.
Fuse used for the round	60 ft.

Ordinary Tamping.—The same procedure was adopted as in the above, but as the cut had not come away clean, it was again charged up, four extra plugs being used for re-blasting the cut. The re-charged "cut" and round were then fired together; 70 = 38·2 lb. plugs being used in all.

Temperature of level	25° C.
Fuse used for the round	66 ft.

APPENDIX III.

Experiment No. 8.—Four cut holes; 26 plugs $1\frac{1}{4}$ in. blasting gelatine = 14·4 lb.; 24 ft. fuse.

Experiment No. 9.—Four cut holes; 26 plugs $1\frac{1}{4}$ in. blasting gelatine = 14·4 lb. electric fuses.

Experiment No. 10.—Three cut holes; 24 plugs $1\frac{1}{4}$ in. blasting gelatine = 13·3 lb.; electric fuses.

Experiment No. 11—Four cut holes; 26 plugs $1\frac{1}{4}$ in. blasting gelatine = 14·4 lb.; 24 ft. fuse.

APPENDIX IV.—(Fuse Experiments).

Experiment No. 1.—13 in. of fuse, weighing 6·887 gm. (dry) were burned in a closed bottle of 12,130 c.c. capacity, and analysed the following day.

Analysis, Calculated on Dry Gases.—

	Vol. %.
Oxygen	17·64
Nitrogen (by diff.) ...	74·31
Hydrogen	1·57
Carbon monoxide ...	2·56
Carbon dioxide	3·51
Olefines	·41
Oxides of nitrogen ...	nil

From measured increase in pressure, as indicated by a manometer, the amount of gas formed was found to be 874 c.c. (N.D.P.). The analysis of the air contained in the bottle before the experiment showed (calculated on dry) :—

	Vol. %.
Oxygen	20·876
Nitrogen	79·084
Carbon dioxide	·04

The saturation with moisture was 60%.

From the above data the composition and amounts of gases given off by the burning fuse were found to be.

Composition, Calculation on Dry.—

	Vol. %.
Nitrogen	22·41
Hydrogen	15·23
Carbon monoxide ...	24·73
Carbon dioxide	33·65
Olifines	3·98
	100·00

Composition in Volume per Cent. calculated to actual underground conditions, *i.e.*, saturated with water vapour at 635 mill-pressure and 25° C.:

	%
Nitrogen	21·58
Hydrogen	14·67
Carbon monoxide ...	23·81
Carbon dioxide	32·41
Olefines	3·83
Water vapour	3·70

Actual amounts of gases in cubic centimetre given off by 100 ft. of fuse calculated to underground conditions, as above :—

	Litres.		Cub. ft.
Nitrogen ...	23·62	=	·834
Hydrogen ...	16·05	=	·567
Carbon monoxide ...	26·06	=	·920
Carbon dioxide ...	35·47	=	1·253
Olefines	4·19	=	·148
Water vapour ...	4·05	=	·143

From the analysis it is also evident, that the fuse in burning consumes a considerable quantity of oxygen, which calculated to underground conditions as above per 100 ft. of fuse, amount to 14·5 litres of oxygen consumed, equal to ·412 cub. ft. Several other experiments on the same lines were made, but the results obtained agree with the above.

The President : I wish to propose a very hearty vote of thanks to the author for his most interesting and instructive paper. I am very glad to learn that this is only an instalment of several papers to come. To mining men and indeed to many others connected with a technical society like ours, the subject must be of the greatest interest, and anything that can be done to improve the state of the atmosphere in mines, confers a benefit not only on miners and the mining community, but upon the inhabitants of the locality generally. The author was good enough to give us an insight into what was coming, in the excellent address which he gave to those members of the Society who went to the Dynamite Factory. I took the opportunity, when there, of thanking the author and the directors of the B.S.A. Explosives Company, for their kindness in having us there and showing us round, and I wish again to extend our thanks to the author and the directors of the company. Many of us had had the pleasure of going there once some years before, and it was very apparent on our second visit that the company had not stood still in its work, but had been and was using every means to improve its processes and manufactures, so that its products might arrive at the highest stage of perfection possible, and I think that is a very gratifying thing to see in a local industry.

Mr. A. McA. Johnston (*Vice-President*) : In seconding this vote of thanks to Mr. Cullen for the excellent paper he has given us to-night, I feel confident that I express the appreciation of all our members. The great amount of trouble he and Dr. Weiskopf have gone to in carrying out these researches is apparent on looking at the figures, and our Society is indeed indebted to them for the pleasure we have had in listening to the interesting conclusions drawn. Whether these conclusions will have the support of all our members is another matter. For example, the figures he has placed before us seem to have on their face some condemnation of the explosives we use here, but as he has most definitely assured us that these are of the best possible quality and surpassed by none, that the most up-to-date methods were adopted in charging and firing, and that every precaution was taken in sampling and analysing the resultant gases, we can only come to the conclusions that a reformation in the constituents of our explosives is essential, that present day methods of tamping need revising and, it would seem, that oil of the best quality for lubricating compressors is a *sine quâ non* for good air. That Mr. Cullen and his able staff are on the track of the former two is a guarantee that all will be done that is possible, whilst the injurious effects of using inferior qualities of oil for compressors is gradually being appreciated by our managers and engineers, as well, let us hope, as by our buyers.

Re ventilation, the author is particularly strong, but this question has so often and so recently been thrashed out by this Society that I feel very diffident about again bringing it to your notice, the more especially as, though the author has given us very alarming figures, he has failed entirely to elucidate their presence, and his data of the actual working conditions are meagre.

A glance at these figures certainly justifies the author's statement that the samples were taken from what are considered the worst parts of the mine, though few of us realised the presence of such quantities of carbon monoxide. The author does well to label many of these dangerous, but I think he ought to have given us a little more light on the probable sources of such large quantities. Given two not improbable premises, that blasting had taken place, say, eight hours previously, and that compressor air was used in driving, we can only conclude that the latter or the gas given off by the broken rock during lashing are the sources of contamination. As, however, lashing should have been finished an hour or two before charging, we are thrown back on the compressor air as our source, or is it possible, the want of it. I think some more detailed information is due to us, and I feel certain the author will be only too pleased to help us out in judging his results.

I am in agreement with him when he says that nitric peroxide must have been formed in every case. Speaking generally, in about a dozen samples of air taken by me some years ago, after explosion, I found this gas in practically all, even though the air samples were taken at a greater distance from the face than in the samples quoted. Until, however, the author and Dr. Weiskopf describe the methods they have adopted in their analyses, I feel we must refrain from undue criticism. In again, therefore, asking you to pass a vote of thanks, I think I can assure the author that his figures will be keenly appreciated throughout mining circles, and we hope that his efforts, and especially his candid avowals, will have the effect of clearing up many points and tend to better ventilation, or it may be to the introduction of more suitable explosives, in these drives whence his samples were obtained.

Mr. C. B. Kingston (*Member of Council*): The author's paper is one of exceptional interest, and I hope it may lead to a full discussion, and bring out some information regarding a number of points connected with the ventilation of our mines, which is not perfect. One point I would like information about, is the amount of gas that may be retained by the broken rock in the face of a drive after blasting. It would be interesting if some of our members could get definite data on this point. Samples of air might be taken a short time after blasting, and again at intervals during the course of lashing, to determine the extent to which the air is affected by the gases retained in the broken rock; and if there is a real advantage to be gained by drenching the rock with water, when practicable, before lashing. The good practice of cleaning out the faces as quickly as possible after blasting is not always followed, and it appears to me that in addition to impeding the flow of air, and obstructing the drive, the broken rock constitutes a reservoir of poisonous gas that is slowly fed into the air the men have to breathe.

Mr. H. A. White (*Member of Council*): I should just like to say that it is the theoretical portion of this most valuable paper of the author which has the greatest interest for those of us who are chiefly occupied upon the surface of our mines. The figures given seem to imply a complete disappearance of about 7 or 8% of oxygen in the gases resulting from the explosion. It is perhaps possible that the solution of this discrepancy would throw some light upon the differences observed in the explosion of gelatine in a steel bomb as compared with the results calculated for the usual bore hole explosion. I do not know if the walls of the bore hole have any special physical effect or if the pyrites present have time to exert any significant chemical action, but the disappearance of so much oxygen seems to require some elucidation.

Mr. C. B. Kingston next read the following paper, first saying, "those of the members who are familiar with the history of gold milling will note that the process described by Mr. Cooke is a return to one of the very early methods of gold extraction."

DESCRIPTION OF ORE TREATMENT AT THE GIANT MINE, HARTLEY DISTRICT, RHODESIA.

By Reginald C. H. Cooke (Member).

In general outline the process consists of crushing with stamps through coarse screens, partial amalgamation, fine grinding in tube mills, close concentration on blanket and canvas tables, followed by amalgamation of concentrates after treatment with nitric acid.

The ore may be described as a heavily mineralised chlorite schist, intermixed with a good deal of iron stone and quartz. The following is an analysis taken from the original report:—

	%
Iron as Fe_2O_3 ...	18·0
Magnetite	14·0
Pyrites	14·6
Insoluble silica and silicates ...	46·0
Al_2O_3	6·0

Specific gravity = 3·1.

The gold is of two kinds, (*a*) that which is easily amalgamated, of a bright yellow colour, 30%, and (*b*) that which is refractory, of rusty appearance, requiring a special treatment before amalgamation. The ore, after passing two Blake rock breakers and falling into the bins with fines that pass the grizzlies, is fed into the batteries by Nelson feeders.

The following table demonstrates the work accomplished by the stamps:—

TABLE "A."

No. of Stamps.	Average monthly tonnage.	Duty.	Weight of Stamps. lbs.	Drop.		Screening.		Discharge.
				No. per minute.	Height. ins.	No, 1 & 2 Batteries.	No. 3.	
15	6,300	15 (of late)	1,250	103 (96 of late)	$7\frac{1}{2}$	$\frac{3}{8}''$	$\frac{1}{2}''$	3″

It may be of interest to mention that $\frac{1}{16}$ in. width slotted screens were originally used with a duty of 10·5 tons per stamp per 24 hours. A $\frac{1}{8}$ in. mesh wire screen was then tried, but unsuccessfully, owing to rapid clogging; $\frac{1}{8}$ in. slotted screens were then ordered, but their arrival being long delayed, at the suggestion of Mr. Galbraith, foreman amalgamator, a $\frac{1}{4}$ in. wire screen was tried with success. Experiments were then made with a $\frac{3}{8}$ in. and $\frac{1}{2}$ in. mesh with the following results:—

TABLE "B."

Screening used.		Approximate duty per stamp.
$\frac{1}{16}$ in.	Slotted screens	10·5 tons.
$\frac{1}{4}$ in.	Mesh wire screening	11·65 ,,
$\frac{3}{8}$ in.	,, ,,	13/14 ,,
$\frac{3}{8}$ in. & $\frac{1}{2}$ in.	,, ,,	15 ,,

It has not been found advisable to use $\frac{1}{2}$ in. screening on more than one battery at a time, except when the ore coming into the mill contains less quartz than usual. If used on all three batteries, the ore crushed by the stamps exceeds the capacity of a single tube mill.

Table "C" shows the result of crushing through a $\frac{3}{8}$ in. *versus* $\frac{1}{2}$ in. screen

It will be seen from this table that the $\frac{1}{2}$ in. screen gives an increase of 12·5% + 30 mesh. As the battery is simply a primary crusher, this is an important factor. The pulp from the batteries passes over copper amalgamating plates 4 ft. long, where 30% of the gold is retained.

N.B.—(The big tonnage and the coarseness of sand passing over these plates does not in any way injuriously affect their surface.)

TABLE "C."

	Battery No. 1. $\frac{3}{8}$ in. mesh.	Battery No. 2. $\frac{3}{8}$ in. mesh.	Battery No. 3. $\frac{1}{2}$ in. mesh.
$+\frac{3}{8}$			= 3·12
$+\frac{1}{4}$	= 2·50%	= ·62	= 7·22
$+\frac{3}{16}$	= 7·38%	= 5·62	= 10·22
$+\frac{1}{8}$	= 6·15%	= 5·00	= 8·00
+ 30	= 25·00%	= 22·50	= 25·00
+ 60	= 12·50%	= 13·78	= 12·45
+ 90	= 8·19%	= 10·00	= 7·75
+ 120	= 5·00%	= 5·62	= 5·21
− 120	= 30·78	= 36·00	= 20·00
	97·50	99·14	98·97

N.B.—The difference between No. 1 and 2 batteries sizing tests is due to the position of the grizzlies, which deliver most of the fines to No. 2 battery.

The pulp then flows down launders over very simple and effective amalgam traps, which prevent any amalgam escaping into the tube mill. These traps consist of a few $\frac{3}{4}$ in. sized pebbles, scattered down the length of the launders, with a false bottom consisting of strips of $\frac{3}{8}$ in screening cut to fit the width of the launder laid over and nailed down. A piece of wood $\frac{3}{8}$ in. square nailed across the width of the launder completes the trap, This allows the amalgam to sink through to the bottom of the launder, where the pebbles, etc., effectively retain it. On an average 400 oz. of amalgam are monthly cleaned up from these traps, and usually no trace of mercury can be seen in the blanket concentrates. The pulp now flows into the tube mill dewatering spitz 4 ft. × 4 ft. A 1 in. nozzle allows coarse sand to sink through the spitz into the tube mill with a ratio of 0·8 to 1 of water to 1 of dry sand.

Table "D," which illustrates the crushing powers of the tube mill, shows a reduction of 41·47% + 30 to 1·56% + 30, and a total of 68·54% + 120 to 22·68% + 120. 180 – 200 tons of dry ore per 24 hours was passing through the tube mill when test was made.

N.B.—The tube mills are 25 ft. long and 5 ft. in diameter, lined with silex blocks; life of liner, 31 weeks; consumption of French imported pebbles, 300 lb. daily; charge, 12 tons; speed, 28 rev. per min.

The overflow of the tube mill dewatering spitz consisting of surplus water with slime containing only about 2% of oversize + 120, is conveyed by a 3 in. pipe to the outlet of the tube, where it joins the pulp discharged from the tube in the usual way. This alone is not sufficient to dilute the pulp to the consistency necessary for obtaining the maximum extraction on blanket tables, and water is there-

Table "D."—Sizing Test.

Sand entering Tube Mill.	Discharged from Tube Mill.
$+\frac{3}{8}$ in. = 1·02%	
$+\frac{1}{4}$ in. = 3·65%	
$+\frac{3}{16}$ in. = 6·97%	
$+\frac{1}{8}$ in. = 6·00%	= 0·00%
+ 30 in. = 23·83%	= 1·56%
+ 60 in. = 13·54%	= 2·00%
+ 90 in. = 8·33%	= 8·00%
+ 120 in. = 5·20%	= 11·12%
− 120 in. = 30·79%	= 76·00%
99·33%	98·68%

fore added by a $2\frac{1}{2}$ in. pipe to render the pulp 7 water to 1 dry sand. This pulp then flows over blanket tables upon which it is concentrated.

Description.—The blanket tables consist of 12 strakes (2 of which are spares) 12 ft. long by 2·5 ft. wide, with a 12·5% fall, these conditions being found to best suit the ore. The blanketing is made to special order in England. In texture it is similar to the ordinary military rug. It is made in long rolls, 3 ft. wide, and is torn up into $4\frac{1}{2}$ ft. lengths, this being the most convenient size for washing. These short lengths of blanket are laid on the strakes like shingles with an overlap of 6 in. The blankets from the strakes are washed at short intervals. When they are to be changed, the pulp is first switched on to the spare strakes. The blankets with concentrates attached are then folded, with the edges always towards the centre, and are carried to the washing vats, washed blankets being kept ready to replace them at once. The pulp is then deflected from the next strakes in series, and so on in regular rotation. In this way each blanket is changed and washed every half hour.

Washing.—The washing vats are 5·5 ft. in diameter by 2·5 ft. deep, filled with water. A boy catches hold of a blanket by the ends and draws it rapidly backwards and forwards through the water; about two dozen motions are sufficient to free the blanket of all concentrates and gold. I am convinced that the high extraction of 55% of total gold recovered on these blankets is due to the above method of using convenient short lengths of blanketing, and the removing and hand-washing of same at short intervals. No amount of sluicing or use of revolving blanket tables would free the blankets of their heavy concentrates or approach the results now obtained on this ore. By the above continued process of frequent changing and washing, a fairly large amount of concentrates—about four to five tons per day—is collected, and as this is too great a bulk to be conveniently dealt with, an elevated blanket table is used for their re-concentration. The concentrates are gradually fed into a perforated box, where a $1\frac{1}{4}$ in. stream of water is allowed to slowly wash them down the re-concentrating table. The overflow from this table falls into the tube mill discharge launder, and again travels over the ordinary strakes. By this method a very rich concentrate of about half a ton is obtained. This is the concentrate from the total day's tonnage, and contains 55% of the assay value of the gold.

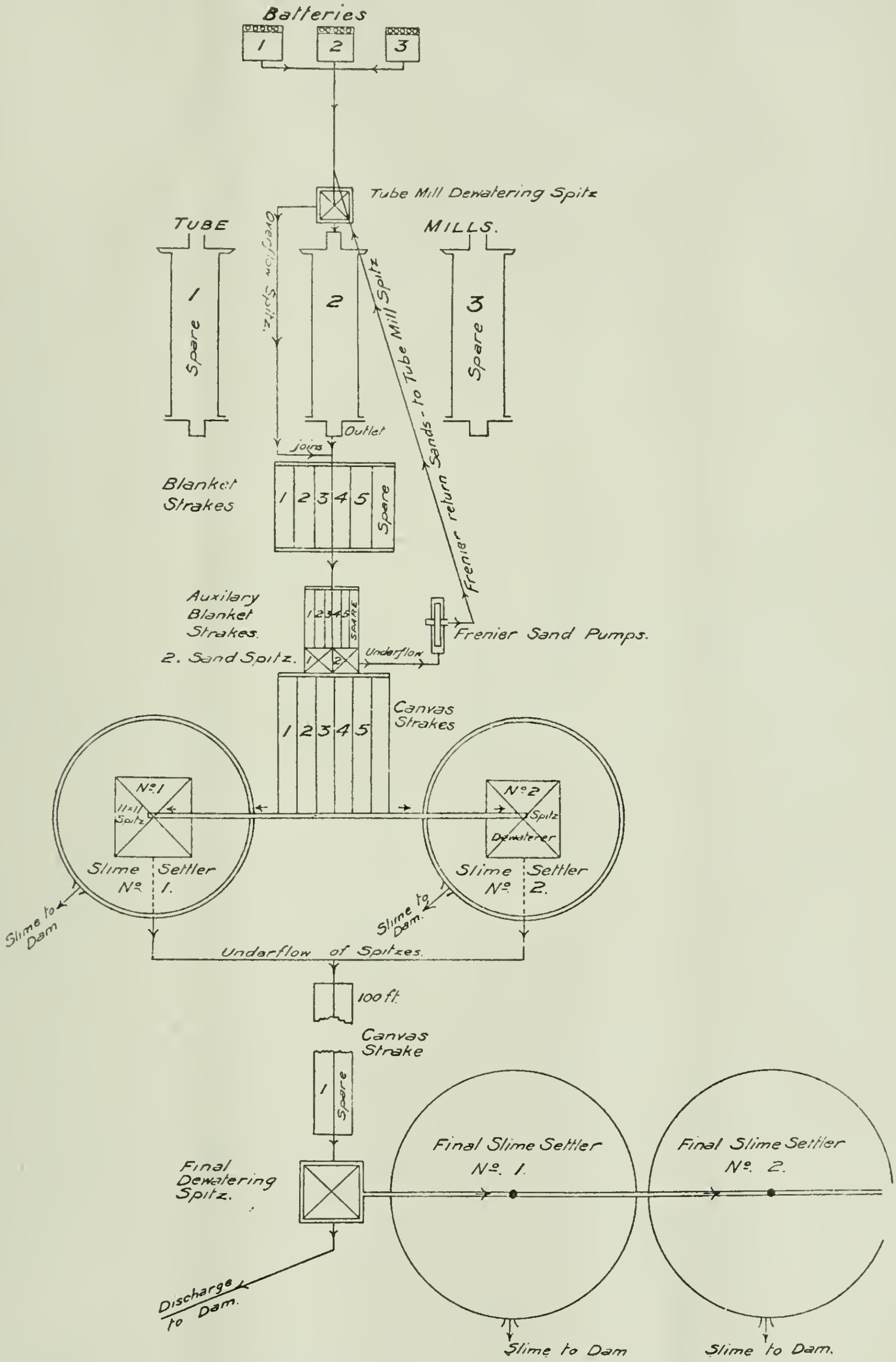
Amalgam Barrel Treatment.—This half-ton of re-concentrates containing the refractory rusty gold is now introduced into the amalgam barrels and about 300 c.c. of concentrated HNO_3 is added to the charge, which is given a run of four hours as a preliminary treatment. Mercury is then added, the rusty gold being by this time sufficiently cleaned by the acid (which also renders mercury more active), and another four hours' run completes the process. Without the acid treatment only a small percentage of the rusty gold will amalgamate, the mercury becoming foul and sickened and much of it being lost by flouring. If the barrels are discharged without the acid treatment, and the amalgam collected in an earthenware mortar, this unamalgamated rusty gold can be seen. We are indebted to Mr. W. A. Caldecott for the suggestion of using nitric acid for cleaning this foul amalgam and rusty gold by pestle and mortar. This led to my idea of using the acid directly with the charge, with success. Many substitutes for the acid have been tried but without success. The contents of the barrels are panned before discharging, and in the event of gold showing, the barrel is re-started with a little more acid. The contents are finally discharged on to the ordinary rotary Batea pan, and the amalgam collected and dressed in the usual way.

To continue treatment—The pulp after flowing over the ordinary blanket tables runs over five narrow auxiliary blanket strakes; it then flows into a two compartment sand separating spitz, each compartment $3\frac{1}{2}$ ft. × $3\frac{1}{2}$ ft., where a $1\frac{1}{4}$ in. nozzle on each allows the oversize + 120 to sink into the lower of the two Frenier return sand pumps; this elevates it to the top Frenier, which in turn sends it back to the tube mill dewatering spitz. Again sinking it passes into the tube mill a second time for further reduction to − 120.

Table "E" illustrates duty of pump.

Table "E."—Pulp to Frenier.

+ 30 = 1·5%
+ 60 = 4·8%
+ 90 = 23·0%
+ 120 = 27·0%
− 120 = 40·0%


Batteries
1
2
3
Tube Mill Dewatering Spitz
TUBE
MILLS.
1
Spare
2
3
Spare
Overflow Spitz.
Sands - to Tube Mill Spitz
Frenier return
Outlet
Joins
Blanket
Strakes
1 2 3 4 5 Spare
Auxilary
Blanket
Strakes.
1 2 3 4 5 SPARE
2. Sand Spitz.
1 2
Underflow
Frenier Sand Pumps.
Canvas
Strakes
1 2 3 4 5
Nº 1
11×11
Spitz
Slime Settler
Nº 1.
Nº 2
Spitz
Dewaterer
Slime Settler
Nº 2.
Slime to
Dam
Slime to
Dam.
Underflow of Spitzes.
100 ft.
Canvas
Strake
1
Spare
Final
Dewatering
Spitz.
Final Slime Settler
Nº. 1.
Final Slime Settler
Nº. 2.
Discharge
to Dam.
Slime to Dam
Slime to Dam.

The overflow from sand separating spitz, consisting of the slimed ore, containing only 1 – 2% of + 120, flows over five canvas tables at ratio of 10 water to 1 dry slime. These canvas strakes are 20 ft. long by 2½ ft. wide, and are covered on the same principle as the blanket strakes, but with the best fine white duck canvas. (Coarse canvas is unsatisfactory both as regards extraction and wear.)

On this canvas is retained the fine rusty gold that has escaped the blankets. The saving on the canvas is 0·25 dwt. gold per ton crushed. The short lengths of canvas are taken up and washed in the same way as the blankets, but as only about 5 tons of concentrates are collected in 15 days (because the canvas retains far less bulk than the blankets) re-concentration to render them rich enough for direct barrel treatment only takes place twice a month. This re-concentration takes place on an auxiliary table covered with canvas.

Battery Tails.—A sample is taken at the bottom of the above canvas strakes before the pulp enters a dewatering spitz in which it is split up into a light portion and a heavy portion.

Heavy Portion—The pulp from the canvas strakes is divided, each part flowing into a large 11 ft. × 11 ft. dewatering spitz, placed in the centre of a 30 ft. diameter slime collecting vat with peripheral oveflow. A 1⅛ in. nozzle on each spitz allows the heavy portion, about 180 tons dry ore, to pass out and flow over a canvas strake 100 ft. long, where 2 dwt. per ton crushed is retained. This pulp retains only 2½ water to 1 dry sand. At the end of this canvas table is a final dewatering spitz 8 ft. × 8 ft. A ¾ in. nozzle allows pulp to flow away to dam only retaining 1 water to 1 dry sand.

Light Portion.—Overflowing the 11 ft. × 11 ft. spitz, having a value of only 1 dwt. or under, this is retained in collectors, from whence it is periodically discharged to waste. The peripheral overflow of collectors gives nearly a clear overflow to return water pump. The overflow of final spitz, also containing light slime, is collected in a 40 ft. collector, the clear water being decanted off to filters connected to it through return water pump. The slime in this vat is also periodically discharged to waste.

Extraction.—Our total extraction averages 87·7%, having been gradually brought up to this point. The tails have been lowered to 1 dwt. and under, by untiring application to every favourable condition discovered by investigation. Cost per ton is high, 5s., but I have every expectation that this will now be reduced by from 3d. to 6d. per ton, due largely to the fact that the tube mills, which have proved troublesome and expensive in maintenance in the past, are now in a satisfactory condition for running: also allocation of spares, which has been somewhat expensive of late, has now dropped 50%.

Before concluding this paper, I wish to acknowledge the courtesy of the general manager and consulting engineer for their kind permission to respond to a special request that I should write a paper for the Society on this subject.

The President: I am sure our best thanks are due to the author for writing this paper. It is very gratifying to have descriptions of processes, plants, and mining work in Rhodesia coming forward so quickly as they do. I only hope that our members up there will continue to keep us well informed as to what is going on. This paper is very interesting, as all the treatment seems to take place through concentration and amalgamation. There is no cyanide treatment at all, and the wonderful recovery of 30% of the gold on the plates following crushing through a ⅜ mesh screen is very extraordinary.

Mr. W. A. Caldecott (*Past-President*): In seconding the vote of thanks, I should also like to express our indebtedness to Mr. Kingston for having obtained this paper for us when in Rhodesia.

In regard to the paper itself, I may say that the stamp duty of 15 tons at the Giant mine, which is one of the mines controlled by the Rhodesian branch of the Consolidated Goldfields of S.A, is probably the highest duty of any gravitation stamps in the world. This is mainly due to the soft nature of the ore, the liners in the tube mills lasting some seven months, and only 300 lb. of imported pebbles being required per day. As our President has remarked, experience has developed a mode of treatment which is in many respects a reversion to early practice, for it is hardly usual nowadays to recover the bulk of the gold in an ore by use of blanketings. As the author points out, satisfactory amalgamation recovery seems practically impossible. for not only is much of the gold rusty or coated with iron oxide, but the pyrites in the ore tend to foul the plates, so that in a short time after dressing the plates become inefficient. When the rusty gold in the ore is rubbed up in a mortar with cyanide solution and mercury, it amalgamates very imperfectly Nitric acid is used for enabling the amalgamation to proceed in the barrel, as it is a solvent for oxide of iron, whereas potassium cyanide is not.

Prof. G. H. Stanley (*Member of Council*): There is just one point that occurs to me: we, most of us, know that nitric acid is not so good a solvent for iron oxide as hydrochloric acid. Why not use the latter therefore?

Mr. W. A. Caldecott: Nitric acid, sulphuric acid, and bisulphate of soda were experimented on, but nitric acid appears the best in practice.

A FEW NOTES ON CUPELLING GOLD-LEAD BULLION.

By Geo. Melvill (Member).

As most of the mines now use the pan furnace for smelting, a few notes on the cupelling of gold-lead bullion are submitted in the hope that they will be of some interest to members.

The usual inside dimensions of the oval test frames used on the Rand are :—

3 ft. 5½ in. long by 2 ft. 3 in. wide by 7 in. deep.

A few years ago the tests were made of bone ash. On the Crown Reef the tests are made of mabor, cement and crushed fire brick, in the following proportions :—

Breast.	Body.
60 lb. new mabor	150 lb. used mabor assay cupels crushed
30 lb. White's English Portland cement	75 lb. cement
30 lb. best crushed fire bricks	75 lb. best crushed fire bricks
about 12% water	about 12% water

It is very important that each ingredient used, be passed through a very fine sieve (say, 1,600 mesh), as the finer the material the longer will the test last. The ingredients are thoroughly mixed and sufficient water added to make it fairly damp. If the mixture is twice passed through a 200 mesh sieve an even moisture throughout is ensured. The body of the test is prepared in exactly the same way as the breast, except that used assay mabor cupels crushed are used instead of new mabor. The breast and body are then put separately into the test frame, on top of which is placed a special frame 7 in. deep, so as to allow the whole mixture to be filled in one operation. It is first trodden down with the feet, then tamped with iron rods about 1 in. thick, and finally with the usual tamper about 2½ in. in diameter. The tamping, with two natives, should not take more than ¾ hour. It is a mistake to tamp tests, made with cement, too much, as this causes cracking. When the tamping is finished the top ring is removed, the surface levelled off, and the test carved out in the usual way, leaving the breast 12 in. and the sides 2 in. from the inside of the ring. The depth may be 4 in. with the sides nearly perpendicular. Four litharge holes (1½ in. diameter) are bored, and four channels cut, which are slightly sloped towards the holes. Three of the channels are 1½ in. wide by ¼ in. deep and one 1½ in. wide and 2 in. deep. The latter is used after all the bullion has been fed, in and saves cutting down, or too much tilting, when finishing. The channels wear down very slightly.

It is a great advantage to have six test frames, so that a test is never less than four or five months old when put into use. The test is allowed to set for at least seven days before turning it on its side, and must then be dried very gently. The cuttings out of the test are put through a sieve and used in the body of the next test. The test is supported on the usual carriage (see "The Metallurgy of Lead," by Hofman), which is made to tilt backwards or forwards by means of a regulating screw. When the test is in position the blower pipe is fixed. This consists of a straight piece of 3 in. pipe very slightly flattened at the end, and put in at an angle of about 30°. There must be no bend or hood at the end.

The dimensions of the cupel furnace can be seen in "The Metallurgy of Lead," by Hofman. The roof should be built with a 4½ in. baffle above the test side of the fire bridge. It helps to keep the flame down instead of travelling along the top of the roof. This is a great improvement, as a more even temperature can be maintained with a small quantity of coal. It is a great advantage to have an independent flue and stack, so that the two furnaces, cupel and pan, can be worked at the same time. The feed hole of the furnace must be built large enough to allow two bars, one on top of the other, to be fed in at a time.

When a new test is being used the heat must be raised very gently, and it is advisable to put a couple of bars in the test when the fire is started. When the furnace is hot enough the lead-gold bullion is fed in until the bath nearly reaches the litharge channel. The heat is then raised a little higher than the usual working temperature, the air blast turned on, and when there is sufficient litharge the feed is pushed in gently until the litharge overflows. The feed is regulated by the amount of litharge in the bath. The working temperature should not be too great, just enough to allow the operator to see what amount of litharge is on the bath. Three and a half bars, about 1,000 oz. each, can be cupelled per hour, and with good clean bullion four bars.

After all the lead bullion has been fed in the temperature is gradually raised, and the test is tipped forward by degrees until all the lead is oxidised and litharge blown off. The gold is then subjected to a high temperature with a reduced blast to eliminate as much of the remaining base metals as possible. The test is now lowered and drawn away from the furnace and the gold is taken off before it sets too hard.

The life of tests on the Rand varies considerably. As many as 520 bars, 1,000 oz. each, have been cupelled in one of these tests in fifteen cupellations. This is probably a record. There is very little trouble with the litharge holes wearing back, and very often six cupellations are done through one hole. The sides of the test wear the most but these are patched with fresh test mixture after each cupellation, and then they are as good as new. With the blower pipe, fixed as before mentioned, the pressure of the blast should be great enough to make about ½ in. deep ripples on the surface of the molten metal. The assay value of the litharge is 1½ oz. As a rule, the gold recovered from the cupel agrees within 6 oz. of the gold called for in the lead bullion, the value of the latter being arrived at from a sample taken continuously during tapping of the pan furnace. The fineness of the bars from cupelled gold averages 842.

These few notes are offered in the hope that they will bring forth some discussion on the subject.

The President: I should like to move a vote of thanks to the author for his notes, which form a very welcome addition to Mr. Thomas' paper. Certainly the life of the "test" which the author mentions is very extraordinary. As to the tilting carriage, which he mentions, I believe the introduction of the device into practical use on these fields was due to one of our members—Mr. Q. J. Leitch.

Mr. W. R. Dowling (*Member of Council*): I would like to second the vote of thanks. Mr. Gibson, the smelter at the Knights Deep, after he has blown off the last litharge adds some flux to gather up the little remaining portions of litharge, and blows that off before attempting to take up the plate of gold. This helps to reduce the amount of base metal in the bullion.

Mr. H. Rusden (*Associate*): I should like to ask the author what type of blower he uses, whether it is of the Root's pattern or an ordinary fan? What material is used to fill the 2 in. deep channel when it is desired to run the litharge over the shallow channel? It appears to me to be rather a dangerous practice to put a couple of pigs of lead in the test when the fire is started, as moisture is frequently retained in the bottom for some time after the heat is sufficient to melt lead in the bath. Lead bullion is usually sampled by taking a dip after the slag has all been removed in the pan furnace, with quite satisfactory results; is there any advantage in sampling continuously during tapping? Our thanks are due to the author for bringing to our notice this mixture for test making which has such excellent properties; in the past I have felt satisfied if a test has stood four to six cupellations, now I feel most anxious to give this mixture a trial, and hope the result will be as satisfactory as the author's.

Mr. G. Melvill: It is an ordinary Sirocco fan.

Mr. H. A. White: I see the author quotes 840 as the fineness of his cupelled bullion. I would like him to let us know the amount of silver and base metal present?

Mr. G. Melvill: The silver runs up to 10 or 11%.

Mr. H. A. White: On the May Consolidated G. M. Co., Ltd., we adopt a method for further refining the gold bullion. We add one or two bars of clean lead after the first cupellation is finished and then run this off. This has the effect of bringing up the average fineness to 860 gold and 110 silver, leaving about 30 of base metals in the bullion.

The Secretary next read Mr. Adair's reply to discussion on his paper, as follows:—

THE ADAIR-USHER PROCESS.

(*Read at May Meeting, 1908.*)

By Alfred Adair (Member.)

REPLY TO DISCUSSION.

Mr. A. Adair (*Member*): In reply to the discussion on my paper, I wish to thank Prof. Yates for his appreciative remarks, and all those who have contributed figures or statistics.

Operations are carried on on such a large scale at the Rand, that the otherwise insignificant loss of 3s. per ton in residues, assumes almost tragic importance, because the loss extends to the large sum of between £2,500,000 to £3,000,000 stg. per annum, or about £7,000 every day. In my paper I have shown how some saving of gold is to be effected, but my critics are unable to rise to the occasion.

Supposing we could save, say, a million or so, or 1s. 6d. per ton. In the first place we should have so much more cash put into circulation, enough money to finance a couple of deep levels annually. But the more important consideration is that of making payable mines out of rock, which does not yield a profit now. There is sufficient of such rock in the Kimberley and other series, to add 50% to our stamping or crushing power, and it is only such an expansion as this, which will ever help Johannesburg as a city. It is considerations such as these affecting the welfare of half the white population of South Africa which induces one to work at gold saving problems, and which helps one to overcome the difficulties and the more deadly inertia that confronts inventors. I believe in my work, and because of that belief am prepared to fight for it against all comers. These difficulties, or some of them, are evident in the discussion.

In the discussion on umber it is most exhilarating to have every one against me. Surely there must be something in such opposition ; but when one comes to the examination one finds mere vague opinions, feeble ones, too, and not one with a firm basis. To one and all I reply, that I gave experimental proofs of each of my contentions, and not one of my opponents has been able to disprove or weaken one of them, viz., that umber, properly used of course, dissolves more gold, especially with weak cyanide solution, and dissolves it in less time. Further, that it conserves the cyanide and hinders the formation of prussiate, therefore it also adds to the certainty of good results. Umber will be of greater service when more rapid processes are introduced, but under the present slow processes I have never claimed a difference of more than a few pence per ton, no plant is required at all, and it adds nothing, or next to nothing, to working expenses.

I now reply individually : Mr. J. R. Williams, beyond casting doubts about, has nothing to say. Both he and Mr. White rake up that old bogey, the Omnibus Patent. I confess my surprise at Mr. Williams. As a member of that remarkable body of discriminators, the Patents Committee, he ought to know that a man can no more obtain a valid patent for a list of oxidising agents applied to cyaniding than he could for a list of mechanical devices, which might be applied to, say, æroplaning.

I have replied to Mr. Melvill on the treatment of slimes, and I am also quite willing to demonstrate to him where he made his mistakes, and why he is getting poor residues now.

Mr. White is generally in a pessimistic mood, and therefore amusing. I think, however, he seldom allows his prejudices to run away with him so completely as in his discussion on my paper. With respect to umber, he advances several *reasons*, as he calls them, against umber. Thus : Umber is not interesting, because, as he fondly imagines, it cannot be patented. His dinner cannot be patented. Therefore he ought not to be interested in dinner. Here is another of Mr. White's reasons : "There are only two oxidising agents in use." If there are but two billiard balls on the table, might there not be room for just *one* more ? Another of these reasons of his, is, that because you cannot use the full theoretical amount of umber, you need not use any, and so on, and so on. Mr. White is uncommonly careful about trifles of wording, but shuts his eyes to facts of importance. He has something to say under the next heading which substantiates this.

Mr. Croghan is rather late in the field, but is a "foeman worthy of one's steel;" he comes with armour in the shape of figures. I regret very much that he confined himself to the pessimistic side of the question. I should like to have seen those back figures he refers to, as his present figures are somewhat beside the mark. I do not take his figures as any real guide to actual practice, because he uses quantities out of all reasonable proportion, and entirely omits slimes from his tests. In testing any process, some attempt should be made to get within a reasonable distance of the practical conditions. Mr. Croghan gets a green solution, showing the presence of an alkaline manganate, an agent never occurring in actual practice but which may account for the loss in alkalinity. But under these conditions where Mr. Croghan in his ·4% experiment used five times as much cyanide and almost ten times as much alkali as is common in slimes treatment ; he also uses about five times as much umber, and also a manganate. Even under these conditions Mr. Croghan cannot point to a loss of alkalinity which may not be due to experimental error, ·197 instead of ·215. I should like to know how the manganate was produced. I have tried umber with all strengths of cyanide, and have not once got it. I easily got it by adding permanganate or sodium peroxide—I doubt that umber of Mr. Croghan's very much.

Some few years ago, one of the contributors hinted in the *Journal* at my venerable years and perhaps also ignorance of advances in science. I did not attempt to reply, because I had really forgotten a great deal. Now, Mr. Croghan deliberately suggests inabilities in theory —well, I love my chemicals, and the more I know about them the less inclined I am to lay down laws for them to follow. But to use Mr. Croghan's own phrase, it requires no "genius of theory" to explain these little matters. Mr. Melvill and Mr. Alexander looked in the wrong place for their acidity. Acidity generally comes from the ore or water, and before blaming umber they should have made quite sure that the acidity did not arise there, or from the effect in Mr. Melvill's case of carbonic acid in an unduly strong alkaline solution. In Mr. Alexander's case the presence of HSCN in his solution proved that sulphuric acid, or perhaps hydrochloric acid from chlorine works residues, had got into his slimes, as only a strong acid could replace the HSCN of the sodium salt. With respect to Mr. Croghan's figures, the manganate may have had something to do with the destruction of cyanide. It is known to all water analysts that all nitrogenous organic compounds are decomposed by it, refer to Wanklyn, Chapman, and Smith (*Journal Chem. Soc.*, vol. v. [ser. 2], 1867, p. 445, *et sequitur*. Others merely express opinions, and these I have dealt with.

I now come to the point of discovering the why and wherefore of all this pessimism and objection to umber.

Fact No. I. So far no tests of the process have been made under the strict conditions usually applied to new processes, viz., tests on the same ore, faithful sampling and assaying conducted throughout with strict impartiality by impartial men. It was stated some time ago that "fair and extensive" trials had been made with umber, and at the same time no mention was made of the fact that I, as the patentee, repudiated these trials, in fact, I knew very little about them. In two of them I knew that the men were utterly incapable of undertaking such responsible work, and in another, the man who was in charge did not know for what purpose he was adding the reagent. No proper tests were made, and to an unprejudiced mind the use of the word "fair" in connection with such tests must bear a totally different meaning to the one usually associated with it. I have not the least doubt that this incorrect statement has misled many people.

Fact No. 2. The balances in general use on the field are incapable of ascertaining the gain or loss of 6d. to 1s. per ton as worked under the usual conditions. The ordinary balance can register only to $\frac{1}{15}$ oz. of gold on the assay ton *i.e.*, 6s. 3d. Notwithstanding the fact that $2\frac{1}{2}$ assay tons are quite sufficient to give good figures with a good balance, 5 assay tons or more have to be used with the common balance to get down to 1s. 3d. per ton, and on many mines the results are only taken at that figure. Hence the impossibility of hoping to show a gain of 6d. per ton, unless long times of testing and better balances are used.

The poorest kind of economy on mines is to supply assayers with inefficient tools. They lose more in a month from this cause than would pay for a new balance every month, because preventible losses of gold occur every day and on every mine. The moral of this is that when people do not know what they lose, they lose nothing, and therefore cannot gain anything.

The Usher plant has now been so generally accepted as *un fait accompli*, that no defence on my part is called for, but perfection is not reached in a day or a year, and there are doubtless defects in the plant, some of which I have already stated. The stand pipe system has been generally adopted, because of its cheapness; in construction some £25 per 30 ft. vat is, I believe, about the figure, and the royalty is small. With the stand pipe, an extra 6d. to 10d. per ton is being gained on the Rand, and the question is whether it is best to secure the 6d. or 10d. per ton now, at an expense which is covered by a month's profit (not to speak of the gain in tank space) or to wait indefinitely until something better turns up. Such informations are coming, but it will take time and perhaps a different form of plant, so I am strongly of the opinion that the man who does not get the nimble 6d. or 1s. now, is behind the times.

My personal preference would be for the moving pipes, even at the extra cost, because it will give an extra 6d. per ton, and the liability to choking disappears; and no attempt has yet been made to improve on the original design. Another advantage is the saving in circulation. The experimental work in my paper points the way to success, and no cyanide man should be satisfied to-day, or for a year or two, until he has reduced his slimes residue to ·15 or under.

Mr. White is again in a pessimistic humour with the Usher process. Here he requires perfection right off and, I am told, refuses to try to get his 6d. with the imperfect plant, and would rather wait for perfection. Mr. White's dislike to anything in the shape of novelty is so marked that one feels inclined to work off a few on him for entertainment. In this place he näively expresses his preference for uniform residues; varying results, even if they show a gain on the whole, are horribly disconcerting to this precision. He sees defects. Why? We all see them. Why does he not try to improve on them, rather than to shut his eyes to his own obvious disadvantage?

Mr. Graham follows, but he is really too reckless in his language, as I shall presently again point out. It was not from "conflicting statements" that I did not give results, but literally and truly as stated by me.

Mr. Pearce refers to G. S. Duncan's patent. I can assure Mr. Pearce that neither Major Usher nor myself were acquainted with this patent, but having seen it recently, I can see the difference which made the difference between failure and success. Mr. Duncan's plan is unworkable on Rand ores.

All Sliming.—I used the term to avoid coining such a term as "The New Metallurgy," but it is apt to mislead, and therefore alter it in my case to "a continuous single treatment process," as, at any rate, unobjectionable on the score of grandiosity.

My description was intended to show the principle only, since a full description would have made the paper wholly unwieldy. And as yet I have not completed drawings.

Mr. Pearce's criticism is the best and calls for a full reply. He is perhaps right in thinking me optimistic about sands being ground for 1s. a ton milled. I admit that, as I was also optimistic in '87, but it is not founded on fancy, as I shall show. I also admit Mr. Pearce's right to speak on tube mills, he has many under his control; why did he not give us some figures? They would be really useful. All my critics fight shy of figures. What is there to be afraid of?

Mr. Pearce seems to be overweighted with Rittinger's theories. I greatly admire the simplicity of his conceptions and the clearness of his expositions myself, but they are purely mathematical, and on a subject which is hardly possible to reduce to mathematical terms. Unless supported by practical results they are pure theories and next to useless for practical purposes. Apart from the irregularities in sizes, shapes and hardness of any given rock, which defy accurate calculation, there are the different machines which may be used—percussive, crushing or grinding. There are also the difficulties that percussion may be employed for one grade, shearing or crushing for another grade, and grinding for a third. The problem will never be solved by mathematics only, perhaps not even be guided thereby. It is therefore useless to wait on that account. Mr. Pearce thinks fine grinding will reach an impossible figure, but he is afraid to come to figures. I point to cement works where the requisite fine grinding of a hard clinker is successfully performed at reasonable cost. But to come nearer. In a week's run on the Rand, in the Adair-Cruickshank mill, ordinary battery sands were ground to 80% through 150 mesh, and all through 90. The I.H.P. accurately determined was 15 for each ton of sand ground, including friction (unknown). Since then, 1893, the cost of everything has been reduced, and the total cost would now be for metal ground, oil, attendance and power, less than 1s. per ton of sand ground. So I was not depending merely on tube mill results.

The tube mill commends itself to me on account of its capacity and simplicity and the certainty of greater efficiency in time. Its performance is not bad now. Mr. Graham gave us the last figures for the Rand, omitting decimals 68 of + 60 and 20 of + 90 mesh were reduced in one operation to 11 of + 60 and 27 of + 90 mesh, 50 of the 88 were therefore put through the 90 mesh; evidently it would cost much less to similarly treat the remaining 38 than the original 88. If Mr. Pearce will compare this 50% and allow 25% for slimes washed out he will see that I did not overstate the grinding effect. The figures for this first grinding have been stated to be as low as 4d. on the ton milled, but even if we allow 8d. the total cost should not exceed the 1s. per ton milled quoted by me. If the tube mill will not do it, then we must revert to other mills that will.

No one seems to have noticed that beside the extra recovery in gold the cyanide consumption would be reduced to one-half. The water consumption lowered by one-third to one-half, zinc saved and much labour; against these items of gain, gold, cyanide, water, zinc and labour, comes the sole item of grinding, and the nett profit therefore works out at 1s. 9d. to 2s. or more per ton over partial tube milling, or 3s. or more without the tube mills. It is on this account that I am utterly opposed to Mr. Pearce when he talks about "making haste slowly." So long as we are losing £7,000 daily we can afford to lose a good deal in hastening, as the £7,000 is irrecoverable once away from the plant. We are sure to make mistakes and will never succeed without making them, whether we start now or wait until another million has gone. I would therefore urge him to move now and leave the waiting and the "pros and cons" to those who have no need to hurry.

On this subject Mr. White opines that I have overlooked something, and although it is buried in words, I infer that it is in reference to the time of treatment of fine sands. It is strange that Mr. White makes no suggestion of his own, and his quotation from Mr. Lea's paper conveys no information on this point. It would be indeed strange if I overlooked one of the main advantages of fine grinding and of my process. Most of us are aware that 70% of the gold in coarse tailings is set free by fine grinding. If Mr. White will take the trouble to study the plant, he will see that there are three new conditions introduced which differ from the sands process, two of which at least favour quicker solution than in the sands plant. I hope he will find them. It is a pity that Mr. White is so very conservative.

As to Mr. Graham, I quite envy his gusty language, it is, however, too reckless, as already pointed out.

Mr. Pearce thinks my statements optimistic, but Mr. Graham says they are misleading, and is promptly moved to refute them. Alas, for brave words, for he does not refute them, not even by deputy.

In the next sentence but one, he asserts that Mr. Pearce has pointed out errors in my reasoning based on false premises. Alas, for Mr. Graham. Mr. Pearce does nothing of the kind. Again, I never said that reduction to − 90 mesh *was* all sliming. In another sentence Mr. Graham refers to my "visionary" description of the plant, I admit its vagueness, but then he proceeds to adjudicate upon something which is merely visionary.

I am glad that one remark of his affords me the opportunity of giving precedence and the honour of introducing a continuous process to Mr. E. T. Rand, and also of expressing my sympathy for his undeserved bad luck. I also sympathise with Mr. Graham in his disappointment with my paper, I was disappointed myself. But when I come to reply to Mr. Graham's criticisms and look for some point to reply to, I can only find visionary phantoms of unsubstantiated illogicalities. Nevertheless, his youthful indiscretions have been enjoyable and made one half wish, well, I do not know what.

Mr. Jamieson's Results.—I was surprised that no one came forward with figures to dispute the statement that we are losing £400,000 per annum over and above what is known, by an assay error, not previously recognised. The best method does not show the whole loss, but is much superior to the methods practised hitherto. It is, however, too tedious for ordinary work, but the loss is too important to neglect, and I would suggest that it be reserved for weekly samples.

Mr. Pearce, Mr. White, and Mr. Graham refer to dissolved gold ; why ? I do not know, because it had nothing to do with the results obtained by Mr. Jamieson, as the sample was distinctly referred to as a sample of slimes, not of residues, hence never had contained any dissolved gold. There is not an assayer in practice who does not know about dissolved gold and does his best with it. But the assay of a sample, is a thing apart from the preparation and is not often attended to by the same man. The gold is got in the assay, whether it had been there originally in solution or not.

Personally I had nothing new to offer on the subject of preparing the sample, and therefore held my peace.

The President : We must thank Mr. Adair for his spirited reply to the criticisms on his paper. I will now ask Prof. Stanley to reply to the discussion on his paper.

A LABORATORY COMPARISON OF TUBE MILL PEBBLES.

(*Read at June Meeting, 1908.*)

By Prof. G. H. Stanley, A.R.S.M., M.I.M.E., M.I.M.M. (Member of Council).

With an Appendix on Liners, by M. Weber.

DISCUSSION.

Prof. G. H. Stanley (*Member of Council*) : There has been no discussion on this paper, and therefore a reply is scarcely called for. I would, however, take the opportunity of thanking you, Mr. President, personally, for your appreciative remarks. I regret that none of our members who have been experimenting with different types of liners have come forward with any figures bearing upon their comparative life, but perhaps these will come forward in some other connection. I hoped to have been able to give you this evening the actual length of the life of the liner mentioned in the paper; I have written for the particulars, but they have not yet come to hand. Still I hope it will appear in the next *Journal.**

BATTERY AND CYANIDE GOLD SMELTING.

(*Read at July Meeting, 1908.*)

By A. Thomas (Member).

REPLY TO DISCUSSION.

Mr. A. H. Hartley (*Member*) : The author is fortunate in having only a difference of 0·14% on weight of amalgam between that and the weight of bullion and mercury after retorting. I find there is a constant difference of from 0·5% to 0·75%, or an apparent loss of from 25 oz. to 40 oz. on 5,000 oz. of hard, machine squeezed amalgam, as against the author's 7 oz. I made a series of tests for moisture and found that by drying slowly for 8 hours at a temperature of 120° F., or until the weights remained constant, the percentage difference was almost identical with the per cent. deficiency after retorting in bulk. When drying the samples I used small trays of pure copper, and loose lids of the same metal in order to avoid any risk of loss in transfer and to detect any possible volatilisation of mercury. In comparing this ·75% to 1% of moisture with that contained in any other hard pressed material, it is not abnormal.

The author rightly emphasises the necessity for thorough and careful calcination in connection with the cyanide clean-up. He might also have profitably dilated on the advantages of a thoroughly equipped acid plant. The present method, in many cases, of continually adding H_2SO_4 to a liquid saturated with zinc sulphate is not the most economical one. Nor is the practice of pumping the same liquid, partially diluted, through the filter press, there to be only partially washed, to be recommended. These are only some of the bad effects of an inadequate plant. It is a matter of time, or plant. The time is never available, and the necessary plant not always.

---

* This liner lasted 111 days.

The President: When reading a book called "Australian Mining and Metallurgy," by Donald Clark, B.C.E., some time back, I was struck by the stress laid therein upon the application of refining flux to "unmelted" retort metal, and the necessity for allowing the flux to act upon the metal in that state "since fluxes have very little refining effect upon melted bullion." Clark also recommends in the case of retorting amalgam, which may be somewhat impure, that nitre be placed in the retort with the amalgam, and as the matter may be of interest in connection with the subject of the paper under discussion I will quote the words of the author of the book mentioned. On p. 491, when dealing with the retorting of amalgam, Clark, having regard at the moment to an impure amalgam, says :—

"A small quantity of some oxidising material, such as nitre, may be placed in the retort with the amalgam ; in this case, however, paper, rags, or other organic matter should be excluded. A little nitre, bichromate of potash or other oxidising agent may be added to a slight depression made on the surface of each ball of amalgam."

The method recommended appears to me a very extraordinary one I have had no necessity for trying it, but I should like to know if it has been tried by the author or by any of our members, and whether the result has proved beneficial or otherwise.

Further on pp. 492-3, when dealing with the melting of gold, Clark says :—"When impure gold is melted it is practically impossible to purify it by mere melting, even with the addition of fluxes. Nitre, carbonate of soda, salt and such materials only attack the impurities in the gold to the slightest extent, while the crucible is attacked violently. When retorted gold is dirty looking or discoloured, it should be melted in a clay pot. The spongy retorted gold, if not very dirty, may be fed into the pot with some nitre. a layer of finely ground salt should also be added; the salt itself has practically no action on the gold or impurities, but as it becomes very fluid when molten it prevents the slags from frothing. Nitre should be added from time to time and the temperature kept low. The base metals will be turned into oxides, and when the gold melts it will separate from them. By prolonging the heating before melting, a greater amount is separated than if suddenly melted, for once gold has melted fluxes have but little action upon it. As soon as the nitre ceases to attack the base metals the temperature is raised, the whole lot melted and poured. The slag in this case consists of salt, which is fluid, and stops on the top of the mould, and oxide of potash containing oxides of the baser metals. The slag will easily wash off with water leaving a very clean gold. When the retorted gold is very dirty it should be melted down in a clay pot as before, but when it is necessary to raise the temperature, borax instead of salt should be added, the borax dissolves the oxides of the metals which have separated out and makes a fluid slag, allowing the gold shots to sink into the main mass. Sometimes, if borax is not added, lumps of oxide of copper or other oxides will form a crust on top and prevent all the gold being poured out. By taking care, the most impure bullion may be purified to such an extent that it contains no base metals: that is only gold and silver. It is quite a mistake to think that base bullion can be purified to any extent by fluxing alone after it has once been melted."

In these days here on the Rand our amalgam is so well cleaned and the retorted metal consequently so pure that refining is seldom necessary, but in other districts, and especially where small mines are being worked it is, I believe, the case that a considerable amount of impure material has to be dealt with, and as the handling of such has not been touched upon by the author the probability of the information being of use to some will form my apology to Mr. Clark for quoting from his work, and to the Society for placing the above extracts before it.

Personally, I have almost always been able to get amalgam clean enough not to require any subsequent treatment beyond a little borax soda flux, which has been sufficient to take up any slight dross or earthy matter which may have been present ; but should there have been a dirty lump among the bulk of retort metal, a few pinches of nitre placed on the centre of the bare surface of the molten metal, followed by borax soda flux to take up the nitre slag, has been all that was required, and in the event of sulphur being present from any pyrites left with the amalgam, a good stirring with an iron rod before pouring. Of course, the point which strikes one in Clark's book is, as I mentioned at first, the necessity for keeping the unmelted metal under the influence of the flux, and I should like to know if the author or other of our members who have to deal with bullion melting consider that this is correct procedure. Naturally, dealing with cyanide slimes in a finely divided state is quite another matter.

Prof. G. H. Stanley (*Member of Council*): It is generally assumed that calcination is the operation which is most liable to result in loss of gold, partly by true volatilisation and partly by mechanical carriage in the fume and gas evolved. It would seem, however, that the loss in the former manner is very small, in view of the negligible vapour tension of gold at the temperatures employed, and the very small effect on this constant, of the presence of zinc and other volatile impurities, such as mercury and tellurium, notwithstanding the very common belief to the contrary.

Bearing on this point, I can cite the following sets of experiments :—

K. Friedrichs[1] (J.S.C.I., Apr. 30th, 1903) made many experiments on the volatilisation of gold-zinc alloys, in which the proportion of zinc varied between 8 and 21,000 times the amount of gold present, and temperatures from 1,050° C. to 3,000° C. The most important results, from the point of the present discussion, are, that at 1,070° with 15 times as much zinc as gold, heated for twenty minutes, the loss was only ·28 per 1,000, and with 23 times as much, heated for thirty minutes, 1·6 per 1,000. Losses were only very large at the highest temperatures with abnormal quantities of zinc. With the lower temperature and much less proportion of zinc present in calcination, the loss by true volatilisation can be but small.

Holloway and Pearse[2] (*Mining World*, Feb. 8th, 1908) found by careful experiments on rich telluride gold ores containing mercury, and with others to which cinnabar had been added : that no loss of gold occurred by pure volatilisation, pointing to the fact that the losses which did occur in practice were purely mechanical.

Rose[3] also finds that on cupellation at 1,000° C. in the presence of various volatile impurities, no volatilisation loss can be detected.

The possibility of mechanical loss, however, is much more serious, and a strong draught could undoubtedly cause a comparatively heavy loss. Attention to this point is therefore essential, but convincing experimental evidence, as regards the value recoverable, is still wanting to justify the erection of dust catching chambers.

With regard to volatilisation losses during bullion melting : Rose[4], (*Engineering and Mining Journal*, August 17, 1907) found that pure gold begins to volatilise just below 1,100° C., and that the rate of loss at 1,250° is four times as great as at 1,100° (he says, though, that volatile impurities increase the loss, but of course, these temperatures are considerably higher than in calcination). The volatilised gold condenses almost at once, and is carried forward by the furnace gases, some of it at any rate, being deposited in the flues. Dust from flues at the London Mint carried 1·39% gold, and at the Sydney Mint, dust even from the coping stone of the stack carried 1·46%.

In view of this volatilisation loss a dust collecting chamber was built at the Philadelphia Mint, which cost £180, and which in six months collected gold worth £900, or 0·1 oz. per 1,000 oz. melted, which Rose states is about the maximum loss. Obviously, on a mine, this proportion would seldom be worth recovery, though in mints and similar situations it is quite worth while on account of the much larger quantity melted. I remember a case at Birmingham, too, where it was stated that the dust from some old furnaces and flues, which were demolished, was of sufficient value to pay for a new set.

Obviously, loss by volatilisation is increased by exposing the gold to a current of gases, in the same way as water evaporates more quickly in a draught, hence the importance of keeping crucibles covered.

Richards[1] gives a very interesting study of the volatilisation of gold and silver in the *Electrochemical and Metallurgical Industry*, March, 1908, in which he states that a vapour tension of ·0002 mm. Hg is sufficient to make a metal show signs of volatilisation ; this should be reached with gold at 942° C., *i.e.*, while the metal is still solid—we have a parallel in the evaporation of ice as such—and it has actually been observed (*vide* Rose and others) at below 1,100° C. when its vapour tension is about ·003 mm. At about 1,300° it increases to ·093 mm., quite sufficient to cause slow volatilisation.

Now to apply this to our conditions : Supposing the furnace gases passing over the gold are at 1,300° C., if saturated with gold vapour it is easily calculated that each cubic yard will carry ·34 gr. of gold value ·68d. Of course, in practice, the air does not all pass over the gold, and as a result is far from being saturated, so that the loss is fortunately not as great as thus indicated ; nevertheless, it is a point worthy of consideration when thinking of melting gold in a reverberatory type of furnace, where it is exposed to the furnace gases (as for instance in cupellation), rather than in crucibles, which can be closed by a cover.

I should like next to refer to some experiments of my own in smelting zinc-gold slimes direct, without acid treatment : they are about two years old now, and I had hoped to find time for more work on the point, but this, unfortunately, has been impossible.

On seeing the slimes treated with nitre cake solution instead of sulphuric acid, it occurred to me that the same result might be obtained by fusing direct with the nitre cake ; afterwards raising the temperature to decompose the base metal sulphates formed, and fluxing the oxides thus produced by an acid flux such as borax or silica. I find since, that the first part of the above scheme had occurred to others also, but not, so far as I am aware, the completion of the conversion to melted gold, by this method, in one operation.

---

[1] See this Society's Proceedings, vol. iii., p. 438.
[2] See May *Journal*, 1908, p. 357.
[3] See Jan. *Journal*, 1905, p. 167.
[4] See October *Journal*, 1907, p. 131.

[1] See July *Journal*, 1908, p. 33.

The first sample of slime I experimented with carried about 20% of gold, and with the small quantity available I was able to make seven tests on quantities of 10 gm. at a time, with borax varying from 50 to 100% of the weight of slime and nitre cake from 100 to 400%, the fineness of the resulting beads varying from 756 to 947. The highest value was obtained with a charge of 4 parts nitre cake, 1 part borax, and 1 part slime, and the button was very malleable.

From the point of view of cost, the best result was obtained with 2 parts nitre cake, ½ part borax, and 1 part slime, the button assaying 866·5, and again being malleable. N P. A second series of experiments was made with a much poorer slime, running only about 1·8% gold : on this five tests were made with 20 gm. lots of slime, the borax in each case being 50% by weight of slime and nitre cake ranging from 100 to 250%. The fineness of the resulting buttons varied from 832 to 987, the highest, as before, being obtained with the highest proportion of nitre cake, but with 150% nitre cake the button still assayed over 900.

On the large scale such a process could be carried out in clay lined crucibles or a small reverberatory furnace, the cost per ounce of gold would probably be less for chemicals and fluxes, though probably more for crucibles and fuel, at any rate in the former case, on account of the larger bulk to be smelted; though as some set-off we have calcination omitted. It would, however, apparently offer considerable advantage in these directions, namely, lessening risk of loss and saving of time.

ZINC DUST PRECIPITATION AT CERRO PRIETO.

By Robert Linton (Associate).

(Read at September Meeting, 1908.)

DISCUSSION.

Mr. S. H. Pearce (*Past-President*): The author's paper on zinc dust precipitation is extremely interesting from the point of view of its re-awakening our interest in the subject. It is of special interest to me, because I had the opportunity some four years ago of seeing the process in operation at the Homestake, and was certainly envious of the facility of cleaning up, as compared to the operation as practised on these fields. I made up my mind there and then that it was exactly what we wanted here, so as to save expense in the material used, and labour in handling the gold slimes. Added to that, there was the advantage that a total clean up was made at any period desired, and there was a minimum of loss by theft. To this end, I persuaded our people to put up the money for the experiment, and Mr. Graham of the Geldenhuis Deep to undertake a series of tests to see if it was applicable here. The final results are expressed in the fact that we are still using zinc shavings. It should not be inferred from this that it is finally settled that the process is not applicable here, for, personally, I am inclined to think that the failure of the experiment was perhaps due more to want of knowledge of all the necessary conditions, rather than to the defects of the process itself.

It is natural to assume that zinc should precipitate gold from cyanide solutions as effectually in the form of dust as in the form of shavings, and I am still somewhat dissatisfied at the failure experienced.

I have read the author's paper very carefully, because I hoped to find a difference of condition that we had failed to fulfil, and that there would be some encouragement to try it again. I find there certainly are some essential points of difference, the chief one of which is the strength in cyanide of the solution treated. The author mentions solutions "as low as 0·7% KCy" being amenable and yielding the gold without difficulty. This was also our experience, inverted, as we found that solutions "as high as ·05% KCy" would give satisfactory results, whereas the ordinary slimes solution which runs often below ·01% were very unsatisfactory. In fact, it was just that failure to obtain precipitation from the slimes solution that caused us to abandon our experiment. The reasons for this are, of course, obvious to those acquainted with our methods here, but I have frequently found some difficulty in explaining them to would-be introducers of new methods of precipitation. For the author's benefit, therefore, I may state that inasmuch as we have found it economical and expedient to use fresh "lead coated" zinc shavings for slimes solutions, and when these are no longer "active," by being fouled by any deposit from this weak solution, they are transferred to other boxes used for the stronger solutions from the sand plant, and become practically as effective as new or fresh zinc for that purpose. It naturally follows then, that this is the weak link in the chain, and any system should be tried primarily upon these solutions, otherwise there would be no means of disposing of the poorly coated zinc, except by means of the acid tub, and considerable expense incurred in the use of both acid and zinc.

I would therefore, ask the author to consider if the conditions we have, could be met, so that the adoption of "dust" here would prove an economic success. I do not think the cost of refining would be a drawback to the process here, seeing that it would be readily run down in a lead furnace, and probably quite as easily as our present zinc precipitate.

As far as our trials of this method at the Geldenhuis Deep are concerned, it is hardly necessary to describe them in detail. The plant consisted of two 10-ton tanks worked alternately. The zinc dust was added as an emulsion to the solution therein and agitated with compressed air for variable periods, as well as during its being pumped through the press. The filter press used was a Johnson 24 × 24 in. gold clean-up press. The trials consisted of varying the amounts of zinc dust, with and without lead acetate and also added cyanide, and passing the solution through the press at varying rates. Defects in the method of filling the press were detected and remedied and, in short, the method was tried under every condition which we could think of, in order to obtain commercially successful results. In looking over the matter, there is one thing that was certainly neglected, viz., the testing of the zinc itself. This may perhaps be of more importance than was considered at the time, but I take it that, however impure it may have been, the use of a larger quantity should have rectified matters, which was not so in result.

The chief reason for the abandoning of the trial was because we found that unless cyanide was added at a ruinous rate to this dilute solution, the precipitation of the gold contents was only effected at such a slow rate as to put the process out of all practical adoption, viz., at 0·5 ton per sq. ft. of filter per 24 hours at most.

The conditions under which satisfactory precipitation is obtained by our existing method are as follows:—

Tons precipitated per 24 hours, 800.

Cyanide strength leaving filters, about ·01%.

Lbs. of cyanide added per 24 hours, 80 lb.

Cyanide strength of solution leaving boxes, about ·008%.

Gold contents of solution entering boxes, 0·66 dwt. per ton.

Gold contents of solution leaving boxes, 0·02 dwt. per ton

The consumption of zinc in precipitating slime solution is not estimated for reasons already given. Taking it over the whole plant, it works out on an average to a little over ·25 lb. zinc per ton treated, irrespective of the grade of the material treated. If expressed in relation to the gold recovered, it would vary from 1 to 3 lb. of zinc per oz. recovered. These figures are considerably in excess of those quoted by the author, and it would be interesting to know how much of this difference is due to the value of the material treated, and how much to our system of precipitation? Perhaps he will supply us also with these figures, so that we can make a comparison on parallel lines.

The author certainly deserves our thanks for his paper, and it is to be hoped that others will follow his example in that part of the world, for, notwithstanding anything that may be understood to the contrary, we are only too pleased to take tips from our fellow workers abroad and to reciprocate in kind.

SMALL MINES OF RHODESIA.

(Read at September Meeting, 1908.)

By B. I. Collings (*Member*).

DISCUSSION.

Mr. C. B. Kingston (*Member of Council*): The author is to be congratulated upon having contributed a paper of great interest and value. The small mines of Rhodesia are becoming important factors in South African mining to-day.

Rhodesia has afforded a welcome outlet to many of the energetic and enterprising men of our community, who have been forced out of their positions by the recent depression. To these have been added other men, of whom there are many in every mining camp, who are attracted by the chances of sudden fortune, and with whom the security of a fixed salary weighs as nothing against the charm of independent action. These men have found an attractive field for their energy in the Chartered Territory, and Rhodesia has been given an opportunity to rehabilitate a mining industry possessed of great intrinsic possibilities, but which was languishing in neglect as a result of the mistaken methods of the pioneer mining companies of the fields, which committed the perhaps natural blunder of attempting to deal with its widely differing ore deposits, on the conventional lines of the Rand. The difficulties into which many of the early Rhodesian companies were plunged may be traced directly to this cause.

When working capitals had been spent in exploring ore deposits that were not quite clearly understood, and as the good chutes had not been developed sufficiently to expose satisfactory reserves of ore for milling purposes, shareholders became discouraged, and directors, in the absence of further capital, were driven to letting the mines on tribute in order to keep the enterprises alive. The early tributors secured very favourable terms and made handsome profits. In addition to having main shafts sunk and equipped, and hundreds of feet of driving done, they, in some instances, had complete milling plants placed at their disposal, and even found ore ready for stoping. They did more or less development work according to the length of their lease. They preferred to do rather less than more, so long as there was ore in sight. Under these circumstances their success was immediate, and quickly attracted the attention of others on the look out for openings. When the available tributes had been absorbed, attention was given to new properties. This led to the development of the numerous individual enterprises included by the author under the general heading, "The Small Mines of Rhodesia."

The extent to which general mining development has grown in Rhodesia will be understood, if it is realised that there are now about 200 producing properties contributing to the published returns of the Government Mining Department. Seventy-five per cent. of these may be included under the heading of "small mines." The balance being operated by companies.

Of the "small mines," some are rightly so called, because they contain a very limited tonnage of pay ore, others are included because of the limited scale upon which they are being worked. Among the latter are some that may be expected to expand into very important enterprises in course of time.

The total output of the Rhodesian mines has already reached a large figure, and would have attracted even more attention than it has, if it were not overshadowed by the phenomenal returns of the Rand. An output of more than £200,000 per month is by no means insignificant, although not more than $\frac{1}{10}$ of the Rand production.

Having spent many years in active practice in Western Australia, the writer has been struck by the many points of similarity between the two fields. Nearly every deposit seen by him in Rhodesia has its counter part in Western Australia.

Rhodesia, it is true, still lacks the Golden Mile of Kalgoorlie, but there is still room in the country for the discovery of a group of mines worthy of comparison with that magnificent area. The pioneer in Western Australia was the prospector and small miner. The "small workers" are doing over again in Rhodesia what was done by their fellows in Australia. These men repay study. They are filled with an abounding energy, and are possessed of an inextinguishable optimism. A little success makes them self opinionated to an almost incredible degree, and their fads are frequently elevated to the dignity of profound theories. Their disregard for accepted theory is not altogether without advantages, because it sometimes leads to the discovery of ore deposits that might be overlooked by a too rigid scientist. The reverse is perhaps equally true. They are for the most part content to deal with facts as they find them during the day and to leave the consideration of theory for the camp at night.

There is probably more technical knowledge among the Rhodesian prospectors and miners than has been the case in other countries, because so many come from the great training ground of the Transvaal. The knowledge gained here, however, must be used with care in connection with the Rhodesian ore deposits, and there are many failures awaiting the man who is careless in transplanting his Rand experience to the Rhodesian field. They soon learn wisdom, however, and men remember the lesson taught by mistakes for which they have to pay out of their own pockets.

The "Small Mines of Rhodesia" represent one of the most interesting of recent South African developments. The men engaged in the work are doing good service for the State. If the motive power of the individual is self interest, the class as a whole, is proud of the work they have done in advancing the development of the country and in disclosing some of its great possibilities.

The author has contributed the first of what may be hoped to prove a series of papers on a subject of great interest. He has broken new ground by his general paper. There remains much to be done in collecting detailed descriptions of the different types of deposit that are being exploited by the "small worker."

The biggest worker of Rhodesia was he whose name the country bears, and his prophecies as to its mining possibilities are being rapidly fulfilled by the "small man," who in the end may prove to be bigger than he looks.

I hope the members of our Society, resident in Rhodesia, may bestir themselves to supply our records with papers and notes covering this fruitful field that lies ready to their hands.

The meeting then closed.

Obituary.

The following deaths are recorded with much regret :—

Mr. J. W. Kilby, assayer of the South Randfontein G. M. Co., Ltd., who died suddenly on the 21st October, 1908. Mr. Kilby was admitted an Associate in February, 1903, and was transferred to membership on the 20th August, 1904.

Mr. Joseph Henry Combellack, mine captain, who died at the Roodepoort Central Deep Mine, Roodepoort, on the 4th November, 1908. Mr. Combellack was elected a Member of the Society in March, 1903.

Visit to Dynamite Factory.

A party of about 70 members visited the Dynamite Factory of the British South African Explosives Company, Ltd., at Modderfontein, on Saturday afternoon, the 24th October, 1908.

On the arrival of the special train about two o'clock, the party were received by Mr. W. Cullen, the general works manager, and other leading officials, and shown over the extensive works and its departments, general laboratory, explosives factory No. 1, box factory, dope factory, tradesmen's workshops, flour mill, sulphuric acid (contact and chamber process), nitric acid, concentration of nitric and sulphuric acids, denitration, lead pipe making and collodion cotton blending, under the guidance of Messrs. W. Cullen, Bachmann, Colquohoun, McIntosh, Lyall, W. Colbert, Donaldson, Campbell, Dr. Weiskopf, W. Waters and G. F. Ayers.

The following is a short description of the factory and its manufactures:

The factory is at present manufacturing about 220,000 cases of explosives per annum, but it is capable of producing 400,000 cases without any addition to the existing plant. The number of people now employed is 240 whites and 700 natives. All the officials and employees live on the property, which is 5,193 acres in extent.

The explosives now being manufactured are designated as follows:—Blasting gelatine, gelatine dynamite. gelignite, dynamite No. 2 (various grades) and coal dynamite. The first three belong to the so-called gelatinous class, and the latter two, as their names indicate, are modifications of the old kieselguhr dynamite (generally called dynamite No. 1), which is not made in South Africa. The basis of all five is of course nitro-glycerine, of which coal dynamite contains only 20% and blasting gelatine 93%. The manufacture of further varieties is contemplated.

Sulphuric acid, nitric acid, and hydrochloric acid are also manufactured, but while the first two are incidental to the making of nitro-glycerine, the last is not. All can be supplied chemically pure.

The company also manufactures lead pipes up to 3 in. diameter, and will shortly be in a position to make sheet lead of any thickness. The manufacture of various other chemical articles for which there is a demand in South Africa will be undertaken at an early date.

The manufacturing operations are highly technical, and the company employs 12 chemists, several of whom are constantly engaged in research work. An immense quantity of water is used in the various processes, and this is pumped from either of the three dams, the total capacity of which is 250,000,000 gallons. The main supply is now being obtained from dam No. 3 abutting on the farm Frankenwald.

At the conclusion of the inspection the party was entertained at a conversazione at the casino. During the course of this function the President (Mr. R. G. Bevington) proposed and Mr. A. McA. Johnston (Vice-President) seconded, a hearty vote of thanks to the company, the general works manager and the staff for their courtesy, kindness and hospitality, and alluded to the great improvements and the extensions to the works which had been carried since the last visit.

Mr. Cullen in responding gave an interesting résumé of the paper subsequently read at the November meeting, which appears in full on pp. 144—151 of this *Journal.*

An interesting demonstration of a new system of shot firing by electricity was also given by Mr. Cullen, Dr. Weiskopf, and other members of the staff.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

The Existence of Platinum Metals in Chemicals used for Assaying.—"During the past twelve months upwards of fifty samples of rock have been assayed in the Government Analytical Laboratory, Cape Town, for platinum, in all but two cases with negative results. These two exceptions having yielded such minute traces it was a matter for considerable surprise when a sample of acid dolerite from Stutterheim in the Eastern Province, and a sample of white quartz from Swellendam, in the Western Province, assayed at the same time, *i.e.*, some four weeks ago, were each found to contain 4·8 gr. of platinum per ton of 2,000 lb.

The possibility of the platinum having come from the rocks appeared to be so remote that it was deemed advisable to test the assay materials in order to ascertain whether they were platiniferous. The hydric sodic carbonate having been proved free from platinum on previous occasions, only the litharge and the borax were examined. In place of possibly platiniferous rocks four assay ton charges of sea-sand were used, and it was then found that when 160 gm. of the litharge previously used were made into a charge with platinum-free borax, a nil result as regards platinum followed, whereas, when 40 gm. of the borax employed in the above assays was coupled with platinum-free litharge, 0·04 mgm of platinum was invariably obtained as the result of the assay.

In such cases the silver bead from the assay showed the peculiar dull spongy appearance which is characteristic of beads containing over 5% platinum. On dissolving this particular bead in hot concentrated sulphuric acid a few specks of a black powder were left which weighed 0·04 mgm. This powder, when dissolved in aqua regia, the solution being evaporated to dryness and the residue taken up with water, gave the platinum reaction with potassium iodide. Lest it should be thought that the platinum could have been derived from the sea-sand an assay was made containing only borax, platinum-free litharge, and sufficient carbon to produce a lead button. In this case, too, the bead produced contained the quantity of platinum already stated. In order to confirm this result further assays were undertaken, one on 200 gm. of borax and another on 1,000 gm. of borax, with the following results :—

200 gm. of borax yielded 0·26 mgm. of black powder,
1,000 gm. of borax yielded 1·42 mgm. of black powder.

The 0·26 mgm., when converted into chloride and treated with ammonia chloride and alcohol, yielded the characteristic yellow precipitate, which, upon ignition, was converted into spongy platinum, and from this was once more obtained yellow ammonic chloro-platinate.

The 1·42 mgm., when treated with aqua regia, did not completely dissolve, but left a residue weighing 0·12 mgm. The residue did not appreciably lose weight after eight hours' treatment with boiling aqua regia, and appeared under the lens as greyish metallic spangles, which were unaltered by ignition. Unfortunately there was not sufficient borax available from which to obtain enough of this substance to determine its nature. Having regard, however, to the foregoing tests and the manner in which the residue was obtained, it seems highly probable that it consists of iridium osmide.

The chloride soluble in water was converted into ammonic chloro-platinate.

The results available, therefore, are as follows :—

40 gm. borax used for assaying dolerite gave 0·04 mgm. platinum.
40 gm. borax used for assaying quartz gave 0·04 mgm. platinum.
40 gm. borax used with sea-sand in place of rock gave 0·04 mgm. platinum.
200 gm. borax used with sea-sand in place of rock gave 0·26 mgm. platinum.
1,000 gm. borax used with sea-sand in place of rock gave 1·30 mgm. platinum and 0·12 mgm. osmiridium.

It therefore appears certain that the borax contains platinum to the amount of 1·3 parts per million, *i.e.*, 0·00013%.

In casting about for the source of platinum it seemed possible that this impurity might have been introduced by fusing the borax in platinum vessels, assuming such to have been used in its manufacture. In order to test this theory 250 gm. of platinum-free borax was fused in a platinum basin at a red heat for four hours. The melt was allowed to cool, and when removed from the dish 200 gm. of it yielded 0·6 mgm. platinum. A second 250 gm. was treated in a similar manner, but in this case so as to obviate the possibility of any platinum having been mechanically removed by the hard fused borax, the latter was poured while still liquid into clean sand. 200 gm. of this yielded 0·46 mgm. platinum. A third quantity confirmed the result obtained in the second experiment.

It is thus shown that fused borax has a very distinct action on platinum basins.

It may be interesting to note that up to the time that platinum was discovered in the borax ordinary assay borax had been employed. The comparatively large quantities of platinum were found in Merck's 'm.g.r. puriss. borax.' Even ordinary assay borax is not always entirely free from platinum, traces of the noble metal having recently been found to exist in a sample at present in use in the Government Laboratory.

Under ordinary circumstances the occurrence of such minute traces of platinum in an assay chemical would not be of more than academic interest, for it would probably not be seriously asserted that rocks containing the infinitesimal traces of the metal which the use of this platiniferous borax would make them appear to contain could pay for working, and as a rule, therefore, time was not wasted in looking for such traces. In view, however, of recent allegations the occurrence of platinum metals in South African rock formations the matter assumes a somewhat different complexion.

The assay balance used in the foregoing experiments turns easily with one-fiftieth mgm., and with care can be made to indicate one-hundredth mgm."—J. G. ROSE, Government Analyst, Capetown, *Cape Chemical Society*, Cape Town, July 17, 1908.—*Chemical News*, Aug. 28, 1908, p. 104. (E. H. C.)

ACETYLENE IN THE LABORATORY.—"For three years Messrs. Erdmann and Makowka have employed liquid acetylene, or the gas, for quantitative analysis of copper, silver, mercury, and precious metals, at the laboratory of the Higher Technical School of Berlin. The copper is easily and completely precipitated by the acetylene in cuprous salts solutions, either ammoniacal or acidified with acetic or tartaric acid, in the form of a granular precipitate, easily washed, of cuprous acetylide C_2H_2CuO."—*Echo des Mines*, Oct. 8.—*London Mining Journal*, Oct. 17, 1908, p. 488. (A. R.)

A HYDROGEN PEROXIDE CELL.—"Two papers have been presented before the American Electro-Chemical Society by H. T. Barnes and G. W. Shearer in conjunction with H. M. Tory and G. H. Cole, in which the results of experiments on the effect of dissolved gases in water on metal surfaces have been given. In the second paper a cell was described which consisted of electrodes of aluminium and magnesium in a solution of aluminium sulphate, to which some hydrogen peroxide was added. Aluminium behaves in a peculiar manner in contact with dissolved air, or oxygen, and becomes electro-negative to a similar aluminium electrode in water free from dissolved air. Magnesium does not show this effect, hence a cell with the two metals for electrodes has a comparatively large E.M.F. developed when dissolved air or oxygen is introduced. The effect is considerably increased by adding hydrogen peroxide. Acting on the suggestion of Prof. Bancroft, tests were applied to the water in which clean aluminium had stood for a few hours, and a measurable quantity of hydrogen peroxide was detected. This was developed from the action of the dissolved oxygen on the aluminium, probably according to the following reaction, also suggested by Bancroft :—

$$2Al + 6H_2O + 6O = Al_2(OH)_6 + 3H_2O_2.$$

The amount of yield of the peroxide was considerably increased in several ways. Without dissolved air no trace of peroxide was observed. The generation of the hydrogen peroxide explains the abnormal E.M.F. developed between two metals so close together in the electro-chemical series."—*Engineering Review*, Sept., 1908. (J. A. W.)

ANALYSIS OF LEAD AND SILVER IN ORES.—"By a series of experiments, Dr. J. Lövy has demonstrated that the Belgian process for dry analysis of lead and silver in ores (melting in iron crucible) is much superior to the English or German methods (melting in clay crucible with addition of metallic iron). The Belgian process gave, with 60 to 65 % sulphuretted lead ores, 98 % to 100 % of the real percentage, determined by the wet method, whereas the yield by the English method was only 92 to 95 %. The percentage of silver determined by direct cupellation of the regulus obtained in the iron crucible corresponded exactly, in the majority of cases, with that obtained by scorification, whereas that of the lead furnished by melting in a clay crucible was always too little. It is to be noted that the deficit in percentage of silver is almost exactly proportional with the loss of lead. It results from this observation, that this deficit is not due to volatilisation of silver, but to the deficiency in yield of lead. The Belgian process, giving 98 % to 100 % can then be employed for estimation of the silver by direct cupellation of the regulus. These data only refer to relatively poor ores (galena, etc.) With those containing a high percentage of zinc the Belgian process also gives a deficiency in results, the greater according to the higher percentage of zinc."—"*Revue des Mines.—London Mining Journal*, Aug. 1, 1908, p. 150. (A. R.)

SILVER IN GOLD BULLION.—"The usual method adopted for the determination of Ag in gold bullion is that wherein the difference between 1,000 parts and the sum of the gold and 'base' contents is assumed to represent the Ag present in the alloy. The gold is directly estimated, and the 'base' determined by loss on cupellation with sufficient lead foil and using a check piece of approximately similar composition. The loss in cupellation will vary according to the temperature and other conditions during the operation, and, necessarily, considerable variations in the results, by different operators, may be looked for. Portions of a well-mixed sample of bullion sawings were submitted for Ag valuation to six independent assayers of repute, and the results advised were :—(1) 105·0 mm. Ag, (2) 99·0 mm. Ag, (3) 104·9 mm. Ag, (4), 97·7 mm. Ag, (5) 102·0 mm. Ag, (6) 104·9 mm. Ag. Two of the assayers, realising the crudity of the method of determination, returned other sets of valuations showing the Ag contents as varying with the temperature employed during the removal by cupellation of the 'base,' an experience similar to our own. In their judgment their particular value in the above table was correct as having been obtained by cupellation at the correct temperature. The other assayers, whilst not submitting the opinion, yet doubtless considered their results as correct by virtue of having performed them at a temperature which experience showed them to yield accurate results, or perhaps results agreeing closely with the results advised upon realisation of current bullion in London or elsewhere. The great variation between the highest and lowest results impresses the difficulty of maintaining a correct temperature for cupellation, and this is to be obtained only by the utilisation of gas-fired muffles with pyrometer installations. Even these are of no utility except there has been predetermined, by actual tests with prepared alloys of known composition, the particular temperatures and conditions for cupellation of bullions of particular compositions. As most mine assay offices are equipped only with coal-fired muffles some difficulty would be experienced in the realisation of that control of temperature necessary to ensure accurate results. Confronted with these drawbacks, Mr. H. B. Wright, chemist of the Great Boulder Perseverance Co., has devoted some time to the problem of working out a wet method of assay likely to yield more accurate results than does the fire method. Finally, the method involving the fusion of the bullion with metallic cadmium, solution of the fused alloy in HNO_3, and titration of the resultant solution with AmCyS has yielded satisfactory results. The triplicate assays of a sample agree in all cases very closely, and as a result of assaying check pieces, alloys of known composition, similar to the material being operated on, the retention of Ag in the undissolved gold is found to amount to about 1 to 2 parts per 1,000. Such amounts of copper as are present in a bullion of composition—Au = 800 mm., Ag = 100 mm, base = 100 mm. —do not interfere with the delicacy of the end-reaction in titration, nor is the method subject to any disadvantages of manipulation, no losses in this respect being possible by virtue of the fact that solution and titration can be performed in the same flask. The method of analysis is as follows :—Standard solution AmCyS.—1·6 gm. of the pure salt dissolved in 1,000 c.c. distilled water. This is standardised against pure silver foil and 1 c.c. equals approximately 4·483 parts of Ag per 1,000. The check piece, containing an amount of silver approximately equal to the contents of the bullion being assayed, is fused with cadmium, and thereafter treated similarly to the actual assays. The approximate amount of silver in the bullion is determined by the usual fire method ; 0·5 gm. of the bullion is fused with 1·5 gm. of cadmium under a cover of potassium cyanide in a porcelain crucible in the flame of a (benzoline) blow lamp. Enough cyanide must be present to cover the cadmium, otherwise the metal will oxidise. A period of five minutes suffices for complete fusion. Allow to cool, place in a stream of running water, when the cyanide will rapidly dissolve and leave the alloy ready for solution in HNO_3. Transfer the bullion to a flask with 20 c.c. H_2O. and add 10 c.c. HNO_3, at a time up to 40 c.c., whilst boiling for one hour, dilute to 150 c.c., add 10 c.c. ferric alum indicator and titrate with the standard solution of AmCyS. An alternative plan, used when time permits, is to allow the buttons to digest in the cold overnight in the flasks with 10 c.c. HNO_3, and then the after-boiling need not be so protracted. Having noted the results produced at different temperatures as judged by the eye, in the cupellation method, and compared the results with those obtained by this wet method, a series of Ag determinations on 14 samples were undertaken (1) by the ordinary cupellation method ; (2) by the wet method. Following are the results in milliemes :—

	Fire.	Wet.
1	127·6	129·8
2	139·3	142·2
3	131·7	134·3
4	146·7	149 0
5	129·4	132·7
6	69·4	70·4
7	58·1	57·9
8	134·3	135·6
9	130·9	132·2
10	138·9	141·8
11	156·9	157·5
12	132·4	135·2
13	141·8	144·0
14	154·1	155·4

As an item of interest it may be mentioned that metallic cadmium can be used as a substitute for silver in estimating the Au in the bullion."—E. H. TAYLOR, *Chamber of Mines Journal*, W. A.—*Australian Mining Standard*, Aug. 26, 1908, p. 235. (A. R.)

ESTIMATION OF GOLD AND PALLADIUM IN SOLUTION.—"A new method to analyse gold and palladium by electric conductivity, invented by Mr. J. Dovan, is briefly described as follows by the *Revue d' Electrochimie et d'Electrométallurgie.* The inventor ascertained, by comparative study of solutions made colloidal by CO, in the case of Au and Pd, that with gold the increase of conductivity is not proportional the percentage of gold in the solution, but that a characteristic formula can be obtained. In the case of palladium, on the contrary, there is a proportion between certain limits. The formula, giving in milligrammes the amount of gold in 100 c.c of liquid, is $z = 1{\cdot}76x - 0{\cdot}0237xy + 0{\cdot}0138x^2 - 0{\cdot}00111x^2y + 0{\cdot}000739xy^2$, in which x is the increase of conductivity and y the initial conductivity of the solution. In the case of palladium, it suffices to multiply the increase of conductivity by the factor of proportion—121×10^2—to obtain the amount of Pd in 100 c.c. of solution."—*London Mining Journal*, Sept. 19, 1908, p. 361. (A. R.)

METALLURGY.

EXTRACTION AND USES OF MOLYBDENUM.—There are two methods in general use for the reduction of the ore :—The alumino-thermic process yields a product free from carbon, but containing small quantities of silicon and from 1 to 2% of iron. Alloys with chromium and nickel are also made by this process. The electrical process is worked by heating the ore in a carbon tube, employing a current of 350 amperes at 60 volts, when a portion of the sulphur is evolved as sulphur dioxide. On increasing the current to 900 amperes at 50 volts, complete fusion is obtained, and the rest of the sulphur is expelled. The metal produced in this way contains about 7% of carbon, of which about 1% is graphitic. The whole of this carbon can be removed by heating the crude metal with molybdic oxide.

Properties.—Metallic molybdenum is of a silver-white colour, and has a specific gravity of about 9. It does not scratch glass, can be readily filed, and is malleable. Moist air does not affect it, but when heated to from 500° to 600° C. in air it burns brilliantly.

The most important outlet for molybdenum, at the present time, is in the manufacture of molybdenum steel. The general effect of the addition of molybdenum to steel, up to 4%., may be said to be to increase the hardness, toughness, and elongation without the production of any deteriorating effect when the steel is heated or welded.

The use of this element in steel manufacture is largely in its experimental stage, and opinions differ as to the value of the alloy in comparison with tungsten steel. Molybdenum is stated to be about three times as powerful in its action as tungsten. Tool steels may contain from 2 to 4% molybdenum, and an alloy containing 3% is stated to be particularly suitable for the manufacture of armour plates. Molybdenum steel at high temperature becomes very hard, but when annealed is softer than tungsten steel. It can be tempered in water without showing fissures, and it is said not to break 'cold short' so easily as does tungsten steel. The molybdenum employed in steel works is usually in one of three forms ; (*a*) a dark-blue metallic powder containing 95 to 99 % of molybdenum ; (*b*) ferro-molybdenum, of which typical specimens have the composition (1) molybdenum 87·5, iron 6·4, carbon 6·3 % ; and (2) molybdenum 75·8 with less than 2 % of carbon, and the rest iron ; (*c*) molybdenum-nickel, containing 75 % molybdenum and 25 % nickel. An alloy with chromium is also made containing 50 % of molybdenum and chromium.

The ammonium salt of molybdic acid is employed in chemical analysis as a reagent for the estimation of phosphoric acid. It is also stated to find a use as a fire-proofing material and as a disinfectant for the upholstering cloth used in railway coaches. Molybdenum salts give a fine blue colour to pottery glazes, and at one time were employed, to a small extent, in the preparation of pigments for textile fabrics. Experiments have also shown that it is possible to employ certain salts of the metal in conjunction with logwood to impart a deep yellow colour to leather.

Commercial Value of Molybdenum and its Ores.—To meet the present requirements of the market molybdenum ores should contain not less than 42% of the metal, and should be free from other metallic minerals. Copper, in more than minute traces, renders the ore practically valueless. The price obtainable, as is the case with most of the rarer minerals, is a matter of special negotiation, and information on the subject is, therefore, not readily secured, but it is stated that ores showing 90 to 95% of molybdenum sulphide (60 to 63·3% of molybdenum) fetch at present from 16s. to 19s. per unit, per cent. of molybdenum, per ton. The price of metallic molybdenum (98 to 99%) is said to range from 5s. to 6s. per lb. Ferro-molybdenum containing 50% of molybdenum is offered at about 3s. 9d. per lb."—*Bulletin of the Imperial Institute.—Iron and Coal Trades Review*, Aug. 28, 1908, p. 873 ; *Page's Weekly*, Sept. 18, 1908, p. 596. (A. R.)

STEEL RAILS.—"The steel rail question has of late attracted much attention in America, and in connection with it many ideas were ventilated at the recent meeting of the American Society for testing materials. Dr. Henry Fay reports the results of microscopical examination of broken rails and classifies the causes of fracture under four heads : (1) unsuitable chemical composition ; (2) segregation ; (3) unsuitable heat treatment ; (4) rolling flaws. Under (1) we have carbon, which may be too high or too low ; phosphorus, which will produce brittleness when in excess ; sulphur, which, if present as iron sulphide, will make rolling difficult ; slag, an indefinite term, embracing oxides or silicates or both. Sulphur, present as manganese sulphide, has generally been considered harmless, but the investigations show it to be extremely brittle and possibly a dangerous material when present in steel under certain conditions. A number of rails which had broken in the foot in crescent form, in nearly every case showed near the top surface of the fractured pieces a thin layer of apparently more brittle material extending in the direction of rolling. Each of these crescent breaks contained manganese sulphide in the form of long threads, and microscopic examination showed streaks of ferrite running parallel to the sulphide. Experiments in producing cracks showed that these usually tend to follow the manganese sulphide lines. In examining rolling flaws, only two fractures have been examined, and it would obviously be unfair to draw definite conclusions from them. However, it is significant that the crack developed in each case followed a streak in which manganese sulphide was embedded. The solidifying point of manganese sulphide is given as 1,162° C. As rail steel solidifies at about 1,450° C., and if rolling begins at any temperature about 1,162° C., the manganese sulphide will be liquid from the rolling temperature down to 1,162° C., and below this temperature the sulphide, being plastic, will be elongated in the direction of rolling. For this reason it is stated that the steel high in sulphur should not be rolled at too high a temperature, as the sulphur is then drawn out into a form that will ultimately lead to trouble. Dr. Fay concludes with the following suggestions. Specifications should be so drawn as to limit the amount of sulphur in steel. At the present time some specifications do not even mention sulphur. The next step is to allow the metal to stand a longer time after the addition of ferro-manganese. The specific gravity of manganese sulphide is 3·966, and that of steel 6·82, therefore if given sufficient time the sulphide should rise to the surface and enter the slag. Usually the time interval between adding the ferro-manganese and pouring the ingots is very short. Dr. Fay concludes by stating that if it is not permissible to start with an initially low sulphur, or to allow a sufficient time interval for the removal of the manganese sulphide, resort must be had to electric refining of molten metal by means of basic slag."—*Engineering Review*, Sept., 1908. (J. A. W.)

The Filtration of Slimes.—"A new filter, the invention of Mr. Burt, of the El Oro Company, is well worth the attention of mining men. Batteries of these filters are in operation in Mexico, and have proved very successful, the features of the machine being simplicity in construction and running, coupled with discharge of slime cakes without manual labour. Briefly, a unit of the plant consists of a steel cylinder, 20 ft. long by 3 ft. 6 in. diameter inside by $\frac{5}{16}$ in. plate, with a door at the bottom which is opened or closed by a toggle operated by the piston of a hydraulic cylinder; this door could be hand-worked if necessary. The filter leaves, of which there are eighteen in each cylinder, are pear-shaped, 40 in. long by 36 in. maximum width, the frames being made of two pieces of $\frac{1}{2}$ in. w.i. pipe, bent and connected at the lower end by a piece of iron rod, and at the upper end to a swivelling water-tight T joint. On the inside of the frame of each leaf $\frac{1}{8}$ in. holes are drilled, with 2 in. to 3 in. spacing, and a piece of cocoa matting is cut to fit the frame and covered by heavy canvas filter cloth sewn on to the $\frac{1}{2}$ in. piping.

The swivelling T joint, which allows the leaf to hang perpendicularly, is connected through the top of the shell to a 4 in. pipe, which serves to carry the solution to its storage tank. Each leaf has a pet cock and valve to allow of the quality of its filtrate being tested from time to time; and, if necessary, any leaf can be put out of action until repaired.

To operate the filter the bottom door is shut, slime under a head of 60 lb. per sq. in. admitted, and the filtrate forced through the leaves and 4 in. header into a storage tank, whence it goes to the zinc boxes. After nine minutes the slime is shut off and the excess slime forced, by means of compressed air admitted at the top of the cylinder, through a pipe at the bottom of the cylinder to the excess slimes tank, the air simultaneously assisting in drying the cakes of slime on the leaves and forcing the filtrate through to the solution tank.

This operation requires eight minutes, after which the door at the bottom of the cylinder is opened and the compressed air switched on to the inside of the filter leaves, which knocks off the slime cakes, so that they fall out of the cylinder into the launder fixed to catch them. A jet of wash water admitted at the upper end of the cylinder clears out the last of the slime cakes and completes the cycle of operations, which totals twenty minutes.

Before admitting the next charge the excess slimes tank is emptied, by an electrically driven centrifugal pump, into the filter.

At El Oro the slime cakes formed vary from $2\frac{1}{2}$ in. to 3 in. thick, and contain about 30% of moisture; whilst the slime coming from the settlers carries 80% moisture.

Each filter has an effective area of 234 sq. ft., and will treat from 120 to 130 tons of dry slime (El Oro) per twenty-four hours. Owing to insufficient filter capacity, the El Oro cakes are not washed with water; such washing would increase the cycle of operations to forty-five or fifty minutes, but the extra capital cost involved would, in most cases, be amply repaid by the additional extraction.

In conclusion, I would add that Mr. Burt has patented the idea, the merits of which should be thoroughly investigated by those about to instal plant; as the first cost is low, much of the tedious process of settling and decantation in tanks is obviated, and, given sufficient fall, but little power is required to operate the plant."—H. G. Payne.—*London Mining Journal*, Sept. 26, 1908, p. 390. (A.R.)

Sampling of Silver-Cobalt Ores.—"There are few ores that present greater difficulty in sampling than the silver-cobalt ores of Cobalt. The ore consists generally of cobalt and nickel arsenides and sulphides, but the trouble is caused by the occurrence of large amounts of metallics composed of native silver, or an alloy of silver and arsenic, which acts in the mill the same as native silver.

The ore leaves the mine in heavy jute sacks containing about 100 lb. each, and is shipped to Copper Cliff, Ont., in railway box cars under seal. In the case of very low grade material no bags are used, and the ore is shipped in bulk.

From the car it is trucked to the weighing scale, where it is weighed in lots of 10 sacks, and the first gross weight obtained. The sacks are then opened and the ore passed through a large Buchanan jaw crusher. The empty sacks are tied up, weighed, and returned to the shipper. If the ore is dry it is shovelled directly into the ball mill. If it is wet it is spread on steam drying plates until it is dry, and then it, too, goes to the ball mill.

As the ore comes from the jaw crusher a small shovelful from each sackful is set aside for a preliminary moisture sample, representing moisture contained in the ore as shipped. This moisture sample is coned and quartered to about 100 lb., after which it is taken to the sampling room, where it is passed through a small Allis-Chalmers laboratory jaw crusher. Then it is cut down to four samples of 5 kg. (11 lb.) each, which are placed in pans in a steam oven, for about 20 hours, at a temperature of about 80° C. This material eventually returns to the crushing floor and goes through the ball mill.

The ball mill is of Allis-Chalmers make and requires 25 h.p. It consists of a large metallic cylinder which revolves horizontally, and is lined with three sets of screens, the finest which is 20 mesh, being farthest from the center. The grinding is done by a large number of hardened steel balls, of a total weight of $1\frac{3}{4}$ tons, which are carried up the side of the cylinder as it revolves, and then drop back on the ore. As the ore is ground to 20 mesh it is discharged below to an automatic sampler. Screen tests show that 50% of the milled ore will pass a 100 mesh sieve, and 80% 50 mesh. The capacity of the mill is about $1\frac{1}{2}$ tons per hour.

The large metallics remain in the ball mill, and after the run is complete, they are removed, weighed, melted in a furnace and run into bars of bullion.

The speiss and the slag from this are combined and sampled together, while the bullion is sampled separately.

The automatic sampler, which is a 27-in. Snyder, cuts out one-tenth of the milled product. It consists merely of a circular casting shaped much like a miner's gold pan, having four openings in its sloping flange, and revolving on the end of a horizontal shaft. Two opposite openings are closed, thus leaving two cuts per revolution. The material to be sampled is directed by a spout so as to fall inside of the sloping flange of the sampler. The rejections slide off the flange and the sample drops through the openings as they pass under the spout. The sampler makes 25 revolutions per minute, and thus gives 3,000 cuts per hour for about 1½ tons of ore, or one cut for every pound of ore, or 60,000 cuts per car of 30 tons. A chain drive prevents slipping so that the cuts are regular.

The main part of the milled product (about nine-tenths of the whole), is here weighed and then passes to the storage bins of the smelter.

The sample is now removed from the sample chamber and weighed, and this weight is added to that of the milled product above. Payment is made on these combined weights, less the moisture.

Two complete weighings of the shipment are thus made which should agree closely. This gives the shipper a check on his weights. Thus the gross weight of ore in sacks should be the same as the weight of (*a*) milled ore including sample ; (*b*) sacks, and (*c*) water lost on drying plates.

A sample for the final determination of moisture is taken by tube sampler from each pailful as it is removed from the sample chamber. This moisture sample is cut down to three samples of 3 kg. (6·6 lb.) each. The result thus obtained is used in the calculation of dry weight. The weight of water lost on the drying plates can be calculated by taking the difference between this and the first moisture result.

The main sample is now thrown on the concrete floor of the sample room, and after being shovelled over twice, is coned and quartered into two halves called sample No. 1 and sample No. 2. These samples are treated alike so that a description of one will suffice for both.

Sample No. 1 is coned and quartered by shovelling on the concrete floor down to about 100 lb., which will be four or five cuts according to the size of the original sample. Cutting down is continued by halving in a Jones sampler till two samples of approximately 20 lb. each are obtained. One of these is placed in a box and sealed by the shipper's agent for future reference, in case any accident should happen to the other samples. The other sample is now dried thoroughly and ground in a Sturtevant laboratory disc grinder till the fines pass through a 100 mesh sieve leaving the metallic scales on the sieve bright and clean. Part of the final grinding is sometimes assisted by a laboratory pebble mill of the Abbe Engineering Co., and sometimes by a Hance drug mill manufactured by Hance Bros. & White.

The metallic scales and fines are weighed and sampled separately. The fines are placed in a pebble mill and mixed for an hour before sampling.

Sample No. 2 is handled as above excepting that no reference sample is retained.

The methods of sampling as described above are according to exceedingly good practice, and the final samples should be about as close to the truth as it is possible to get them.

It requires three days to complete the sampling of a 30-ton car."—ARTHUR A. COLE.—*Mining World*, Aug. 1, 1908, p. 164. (J. Y.)

THE PREVENTION OF RUST.—"A paper on the prevention of rust was recently read by Mr. Thornton - Murray before the Birmingham University Metallurgical Society. The author first dealt with the theories of rust. It was once considered to be due to chemical action, carbon dioxide and water being necessary for its occurrence, and the compound formed having the formula $2Fe_2O_3.3H_2O$. The supposition has to some extent been modified by an electrolytic theory due to Cushman, who considers that the action is mainly promoted by irregularities in the metallic surface, though the presence of carbon dioxide and water greatly facilitates matters. Turning to the prevention of rust, Mr. Thornton - Murray briefly described the various methods employed for this purpose, including tinning and galvanising, and showed that of the different varieties of iron, some resist rusting action to a greater extent than others. In conclusion, he mentioned a method, due to Mr. T. Corlett, of Birmingham, which consists in immersing the article in a hot solution of some phosphate and an iron compound. This covers the metal surface with a mixture of ferrous and ferric phosphates, and presents a pleasing dull black appearance. The process, it is said, makes the iron highly resistant to corrosion, and is adapted for light engineering work, such as cycle frames, gun barrels, stampings, and press work."—*Indian Engineering*, Aug. 15, 1908, p. 109. (A.R.)

NICKEL.—"With the exception of the nickel contained in the ores shipped from the Cobalt district, the production of nickel in Canada is derived entirely from the well-known nickel-copper deposits of the Sudbury district. The output has been increasing steadily for a number of years, athough the actual amount of nickel contained in matte shipped in 1907 is somewhat less than in 1906. Two companies are carrying on active operations: The Mond Nickel Co., at Victoria Mines, and the Canadian Copper Co., at Copper Cliff. The ore is first roasted and then smelted to a Bessemer matte containing from 77 to 80% of the combined metals, copper and nickel, which is shipped to the United States and Great Britain for refining."—JOHN MCLEISH.—Sessional Paper, No. 26*a*, Department of Mines, Canada, Summary Report of the Mines Branch, 1907-8, 1908, p. 73. (C. B. K.)

MINING.

PREVENTION OF COAL-DUST EXPLOSIONS.—"Experiments have been made by Mr. Dautriche, engineer of the Poudres and Saltpêtres, on the means of suppressing flames at the mouths of cannon. This he succeeded in doing by producing a cloud composed of potassium or sodium salt. This cloud opposes combustion of gases and inflammable dust. This discovery has a very wide range. Its application in fiery mines and industries where the dust is liable to cause serious explosions will make it possible to greatly decrease the number of accidents. It appears that the action of sodium carbonate is the same as that of this potassium or sodium cloud. It must suppress or greatly abate the flame usually produced by explosions, and, consequently, permit of the use of much heavier charges of explosives without danger of conflagration."—*Revue Industrielle*, Sept. 26.—*London Mining Journal*, Oct. 17, 1908, p. 488. (A. R.)

COAL DUST AS A FACTOR IN MINE EXPLOSIONS.—"At the June meeting of the Coal Mining Institute of America a valuable paper on the subject of mine dust as a factor in coal mine explosions was read by Dr. Henry H. Payne. Dr. Payne is in a position to speak authoritatively on the phenomena of coal mine explosions, and his paper is made doubly valuable by the fact that it presents a concise review of the more important publications on the subject and a summary of the conclusions reached in the extensive researches on the conditions determining the elements entering into and controlling the inflammability of various dusts, which have been made in England, France and Germany. In conclusion he sums up what his own experience and his wide reading lead him to believe are the established facts and the most creditable theories.

Granting that secondary explosions may be propagated indefinitely by the larger sizes, only coal dust which will pass through a No. 100 screen is capable of initial or primary explosion.

Such dust, after having been subjected to ordinary atmospheric air for only a few hours, becomes largely decomposed and exists as a bubble of constituent gases whose film is composed of undisintegrated carbon and impurities, which, after combustion, are known as ash.

Such dust, when suspended in a homogeneous cloud with a moderate air velocity, is susceptible of ignition either through shock, compression, or sufficient heat to inaugurate combustion.

Such results will be classified as combustion or explosion, depending upon the volume of dust ignited, its supply of oxygen and the space within which combustion takes place.

Where gas alone is ignited, and the mine is free from dust, a 'high explosive' is obtained, and the explosion may be strictly local, due to the cooling effects of the walls,

The chief product of a dust explosion is carbon monoxide, whose expansion under combustion is greater than that of methane, and which receives its continuous supply of oxygen by feeding toward the intake.

A dust explosion, while assisted by the intake air, must nevertheless, follow those entries or airways furnishing the most material upon which to feed; and when this course follows the return airways, the conversion of the carbon-monoxide to carbon-dioxide renders the air extinctive, and prohibits further propagation. Evidence of such a condition will be found in the coke splashing or crusts, formed by the deposit of red-hot cinders carried by the air waves, and testifying to incomplete combustion.

The initial explosion may, and generally does, distil so large an amount of gas that complete combustion is impossible at the site of distillation, and this mass of gas and dust, in varying stages of ignition and combustion at a temperature greatly in excess of the point of ignition, were sufficient oxygen present, will develop into local explosions at irregular intervals wherever adequate atmospheric oxygen is available, such as at junctions of airways, widened passages for side tracks, or cavities where falls of roof rock have occurred, and are frequently called 'flame areas.' When such an explosion, either primary or secondary, travels toward a dead-end of an entry passage, the compression generated by its expansion and momentum causes an almost incredible rise in temperature, sufficient to distil the various hydrocarbons from even the ribs of the coal itself, and supplementing it with a heat potential far in excess of its losses through radiation and expansion.

The liability of any coal dust to explosion increases almost directly with its percentage of volatile matter that is combustible, *i.e.*, the quotient of its percentage of volatile matter divided by the sum of the percentages of volatile matter and fixed carbon.

While coal dust alone, under the conditions enumerated, is distinctively explosive, the presence of even the smallest amount of methane augments materially the susceptibility to ignition.

On account of the great elasticity of air, it is highly probable that no proper conception has yet been attained of the almost incredible speed with which a dust explosion, through its gaseous products, may be extended to far distant portions of a mine under the force of initial expansion, properly called the 'percussive theory.'

Changes in barometric pressure only affect the liability to explosion in so far as they allow, when the barometer is falling, a settling of any possible accumulation of methane from a dome in the roof-rock into the ventilating current; but such small quantities of gas are infinitely safer when diffused in the current of air than when concentrated in one place. Moreover, even if we grant that a low barometer allows greater occlusion, it also allows easier ventilation for the diffusion of such occlusion.

A mine may be overventilated until the air-current has such a velocity that it stirs up dust [and would feed any slight ignition which might take place and otherwise die out.

The difference in the amount of real dust made by either air-punchers or electric chain machines is so slight, and so variable, depending on the nature of the coal and the skill of the machine man, that it cannot be said that either machine, as a class, creates more dust, than the other.

The results of experiments with electric ignition of dust show that the danger from electric wiring is no greater than that of stirring up a cloud of dust from a broken air-pipe or a loose connection.

Coal dust cannot be made wet, in the usual sense. The method of fine spraying is indicative of the best results; but, even then, it is hypothetical if the most careful system of watering is not merely an infinitesimal portion of the 'ounce of prevention,' and it is an open question whether it is not positively detrimental."—H. H. PAYNE.—*Engineering Magazine*, Aug., 1908, p. 773.

EXPERIMENT IN MINE VENTILATION. — "It is interesting to quote the following from the annual report of the Western Australian Mines Department: —'Inspector Cullingworth has forwarded the following description of a rather interesting experiment recently tried in mine ventilation in the Boulder Deep Levels mine:—The 1,100 ft. level of this mine is badly ventilated, there being no winzes down below 500 ft. The conditions are further aggravated by the fact of the shaft not being centred for the last 30 or 40 ft., and being a wet shaft, and the water kept down by baling, there is a continuous shower of water which spreads all over the uncentred portion of the shaft, and further serves to deaden the air. The experiment consisted of affixing to the compressed air column on the surface a locomotive vacuum air brake, and connecting with a steam pipe from the boiler. The compressor was stopped, and the hose taken off the air pipe in the level below. Steam was then turned on, passed through the air brake, and exhausted into the air. We then descended to the level, taking a thermometer. On the plat when we landed the temperature was 82° F. At the face, some 600 ft. from the plat, there was a good current of air rushing up the air pipe, sufficient to put out the light of a candle at 2 ft. In about 10 minutes' time the temperature all along the level to the plat had been reduced to 74° F. So far as I am aware, this is the first time a Westinghouse vacuum air brake has been tried for this purpose. The experiment was merely preliminary, and the question of costs cannot yet be arrived at. It seems as though it would be well worth continuing these experiments in a smoky face after firing, where at present, even with a jet of air on, it takes some hours to clear the smoke.'"—*Australian Mining Standard*, Sept. 16, 1908, p. 310. (A. R.)

Fuse Test.—From time to time in connection with accidents on the goldfields, it has been alleged that samples of fuse have "run through," causing premature explosion. In the course of 13 years' inspection of the fuse of this State, involving the examination of many thousands of samples, I have never encountered one which "ran." It occurred to me, however, that it might be possible that "short circuiting" occurred, due to the method of handling the fuse in our mines.

When a miner fires a "round" of holes, it is customary for him to tightly double up the length of fuse protruding from the holes and force it, in a crushed condition, into the mouth of the bore hole, this being done to prevent the fuse being cut by flying stones, etc., from the earlier holes of the "round." In order to test this, galvanised iron tubes were employed of 1½ in. internal diameter (the size of the ordinary machine hole on our mines), and about 12 in. long, and lengths of fuse 6 ft. long (about the average length of the miners' "stick") were coiled or bent and tightly crushed into these tubes; they were then burned alongside of lengths of the same fuse laid out on the ground in the ordinary way.

The results were very striking. Out of several hundred tests, no increase of speed was manifested by the samples in the tubes, and in fact, as a general rule, by this treatment the rate of burning was retarded, in some cases to a considerable extent. Of the total number of 287 fuses tested in tubes 107 showed a retardation as compared with fuses burnt in the open, while only *two* showed an acceleration of speed, and this very slight.

While this method of treatment, therefore, has no effect in accelerating the burning rate of the fuse, and the tests still further emphasise the improbability of any of our safety fuse "running through" during use, it is of some importance to note that a certain retardation of explosion may be caused by this doubling of the fuse, and give an additional reason for discouraging the tendency, which often exists, for men to return too quickly to holes which they suspect have missed fire.—E. A. Mann.—*Report of the Department of Mines of Western Australia for the year 1907*, p. 221. (A. R.)

MISCELLANEOUS.

A Paint with a new Use.—"It is stated that a paint is about to be placed upon the market which will serve as an indicator of excessive heat in machine parts. When the parts painted are cool, the paint is red in colour; but when they become hot, the colour becomes black, and returns to red as the part cools again. This paint is said to consist of mercuric iodide and cupric oxide, mixed in varying quantities, according to the amount of heat considered permissible."—*Indian Engineering*, Aug. 22, 1908, p. 123. (A.R.)

Ankylostomiasis in South Africa.*—"The disease ankylostomiasis, commonly spoken of as miners' anaemia, is produced in South Africa by the presence in the small intestine of one, sometimes two, species of nematode worms, namely :—

(1) Ankylostomum duodenale.

(2) Necator Americanus.

It was Dr. Mathias, of Kimberley, who, in 1896, first drew attention to the fact that the peculiar symptoms of anaemia, etc., from which so many of the De Beers European miners were suffering, were due to the presence of the ankylostoma parasites. To Dr. Mathias all credit is due for first sounding a warning to South Africa of the presence of this dangerous disease among a section of its community.

In 1905, while I was examining gangs of native labourers, freshly recruited from the mines, my attention was attracted by the pigmented patches, such as I read Delamere considered pathognomonic of ankylostomiasis, on the tongues of some of them, though I know from subsequent observations that this pigmented condition is common amongst most Bantu races, irrespective of the presence of the ankylostoma parasites. It was this which first led met to make a systematic investigation in the matter.

A microscopic examination of faeces, collected in the native hospital, from patients of various tribes, at once proved that the disease was prevalent among them. I could give the results of some hundreds of examinations of faeces, but I think it unnecessary, as in the hospital I naturally selected the excreta of patients who were anaemic and showed other signs of the disease, consequently the number of positive results obtained from this source was large, and is liable to give an exaggerated idea of the proportion of natives infected.

Geographical Distribution.—If we consider first the natives of the East Coast, we see that the Nyassa, Mozambique and Quilimane boys, that is to say, those who occupy the territory between the southern boundary of German East Africa and the Zambesi, are grossly infected (61·9%).

A little further south, natives from Beira, coming from the old Gazaland, on the banks of the Pungwe River, are shown by the limited figures at hand to be infected to some extent.

Of the inhabitants living in the interior of the country, along this length of coast line, the table shows records of natives in British Central Africa, Nyassaland and on the eastern borders of Lake Nyassa, also natives which I have classed under the heading of Angonis, as most of them belong to a Zulu tribe of that name, and who come from the Zambesia district, about Lat. 16° S. to the S.W. of the southern portion of Lake Nyassa.

A percentage of the inhabitants of all these inland districts harbour the parasite.

South of Lat. 22° S. we find about 35% of Myambaan, Mtyopi, Shangaan, and Tonga boys are infected, consequently the worm is found in the coast countries between Lat. 22° S. and Delagoa Bay.

Of the inhabitants of Zululand I cannot speak from personal experience, but Dr. Hill, Medical Officer of Health for Natal, in his report for 1906, states that Dr. L. G. Haydon demonstrated the ova of the worm in 3·9 per thousand of native prisoners removed to Durban after the last Zulu rebellion.

We must therefore assume that both Zululand and Natal are infected to some extent.

---

* See "Ankylostomiasis: A Warning," by W. C. C. Pakes. This Society's Proceedings, vol. iv., pp. 389—404 (Jan., 1904).

I hesitate to give an opinion as to the presence of the disease or otherwise in Damaraland. The table shows that two Maherero harboured the parasites, but possibly they were infected in the Transvaal, as they had been working in the mines for some time.

As regards the rest of the country, I believe that the high veld and more temperate parts are free from the disease ; that is to say, the Cape Colony (excepting De Beers' mine at Kimberley), Orange River Colony, Basutoland, Bechuanaland, and most of the Transvaal. I make this statement because I have not come across the disease among many thousands of natives passing through my hands from these parts.

The next question to be considered is: 'What effect has the importation of this enormous body of infected labourers on the underground workers in the mines of the Transvaal ?'

Knowing the manner in which the disease spread in Kimberley and some of the mines in England and Europe, one would be justified in anticipating disastrous results.

I think, however, that those responsible for the mines are to be congratulated on the fact that, excepting the underground hands on one (possibly two) properties, the European miners have not become infected, nor as a general rule have the native labourers been re-infected.

In view of the previous statements showing the amount of infected material which must for years have been poured into the mines, the assertion that the European miner does not contract the disease is a daring one to express, and requires substantiation before it can be accepted. I therefore submit the following facts for your consideration :—

(1) I have made careful inquiries from medical men practising on the mines as to the occurrence of the disease among Europeans employed underground. With the exception of men on one mine, I have only obtained accounts of two Europeans being infected. One of these was a recent arrival. I am not certain that the other had not been working in the infected mine already referred to.

The medical men along the reef are thoroughly alive to the possibility of the existence of the disease, and specimens of the faeces of miners suffering from symptoms in any way resembling those of ankylostomiasis are frequently submitted for microscopical examination, and negative results obtained.

If the disease were spreading to any extent among the Europeans, it would be recognised, as it was in Kimberley and on the one infected mine here.

(2) I think a more convincing argument against the infection occurring extensively is that whereas the percentage of infected natives who have never been below ground is 55·48, that of boys who had worked on the mines is 32·35, and most of these have certainly worked barefooted in the mines for months; because, if infection were occurring to any extent below ground, there would certainly be no reduction in the numbers found to be harbouring the parasite. On the contrary, after a period of exposure during their work in the mine, one would expect a marked increase.

(3) While making post-mortem examinations I have been astonished at the small number of parasites found in each intestine.

(4) In the course of my duties I see hundreds of sick natives of all tribes who have worked on the mines, yet I cannot call to mind cases of ankylostomiasis occurring among any but tropical boys. If the mines were seriously infected I should see cases among the M'Xosa and other Cape Colony natives, as well as among those from more northern parts.

These facts have satisfied me that at present the miners on the Rand, European and coloured, are not being infected with the disease, or at any rate, if they are, it is only to a very trivial extent. Whether this satisfactory state of affairs will continue in fresh mines remains to be seen.

We have now to consider, under the circumstances, why everyone employed underground is not infected.

The filthy habits of many of the natives exclude all idea that the sanitation is so perfect that it prevents the disease spreading. Other factors must therefore be looked for. I believe there are several in operation. To the best of my knowledge they are:

(1) The acid reaction in many of the mine waters.
(2) Iron salts contained in some of the mines.
(3) Low temperature of some workings.
(4) Dryness of some mines.
(5) Nature of soil in the mines.

These factors must be considered separately, though more than one of them may exist in some mines or in one portion at the same time.

(1) *Acid Reaction of Mine Waters.*—Many of the mine waters are extremely acid. One sample of a series I had taken showed an acidity equal to 0·79% sulphuric acid. The degree of acidity varies in different mines and in different parts of the same mine, depending as it does on the oxidation of the pyrites and on the amount of water passing over them. Therefore, until the reef is exposed, very little acid is formed, consequently it takes a much longer time for the water of a deep level mine to acquire an acid reaction than for that of an outcrop. The following analyses demonstrate this point :—

	Vertical Depth. Feet.	Degree of Acidity
Mine A.		
Samples taken on 6th level ...	786	1·55
,, ,, 8th level ...	1,049	28·20
,, ,, 10th level ...	1,348	26·76
Mine B.		
Samples taken on 6th level ...	820	2·70
,, ,, 8th level ...	1,086	19·80
,, ,, 11th level ...	1,618	43·30

Mine C.

	Degree of Alkalinity
Samples taken from vertical sump ...	0·5
,, ,, east incline	0·27
,, ,, east shaft tank ...	0·18

Mines A and B were outcrop mines, with markedly acid water. C was the deep level mine to A and B. Its water was alkaline throughout, which may be accounted for by the fact that there was a large amount of water coming down the shaft through alkaline strata. The amount of reef exposed in the mine was small, consequently there was insufficient acid formed to neutralise, much less acidify, the bulk of the mine water. One may expect, however, that, as the mine is worked, and as more and more reef is exposed, the alkaline water will be neutralised and finally given an acid reaction.

This will, I trust, give some of the reasons accounting for the degree of acidity varying in different places.

To return to the effect of the acid reaction on the spread of ankylostomiasis. Dr. Boycott, when reporting on the Levant and other mines, states that the sodium chloride in them prevented the spread of the disease. He found that a 2·5% salt solution quickly kills ankylostoma larvae. If a saline solution is useful in this manner, one would expect an acid one to have an equally beneficial effect. Dr. May, the Government Bacteriologist for the Transvaal, undertook experiments in this matter, which prove that a 0·75% solution of acid stops the formation of ankylostoma larvae, and that probably a much lower degree of acidity will prevent the larvae when formed from penetrating the skin.

It must therefore be admitted that as some mine waters have a larger amount of acid than 0·75%, this acid reaction does at times safeguard the underground worker from infection.

(2) *Iron Salts.*—Dr. McCrae, the Government Analyst for the Transvaal, has suggested that possibly the iron salts (sulphates) may act in the same manner as the sodium chloride has done in some Cornish mines in preventing the spread of the disease, but experiments in this direction have not been made.

(3) *Temperature.*—It is stated that larvae do not develop at a temperature below 77° F. Some of the workings of the mines in the Transvaal do not reach that temperature.

This factor must be looked upon as a further safeguard.

(4) *Dryness.*—The larvae cannot develop without a certain degree of moisture. Some mines in the Transvaal are excessively dry, and so are protected in this way.

(5) Dr. Mathias has suggested that possibly the nature of the soil below ground may have an influence on the spread of the disease. He suggests that a foundation of finely-ground quartz, such as might be found in parts of gold mines, would not be such a suitable medium for the larvae of the parasite to live in as, for example, the moist blue ground of a diamond mine.

We see, therefore, in discussing the presence or absence of this disease on the Rand, one must consider each mine and each portion of each mine on its merits.

Probably other causes will yet be discovered to account for the non-infection of the miner of the Transvaal.

I mentioned one mine as being an exception. On this mine, every European employee who went below ground, from the manager downwards, contracted a cutaneous eruption, recognised as 'bunches,' and was found after a time to harbour the parasite, and had to undergo treatment.

The mine in question is a deep one; the temperature below ground was high; it contained a large quantity of water, samples of which gave a uniformly alkaline reaction when tested. The underground worker was therefore not protected by any of the four factors described above.

From all accounts it appears that the timbering in the mine was the most fruitful source of infection—miners sitting on damp wood contracted sores on their buttocks.

I have noticed that many pneumonia patients from tropical parts develop an irregular temperature on or about the fourth day after their crisis.

This irregularity is, I believe, often produced by the parasites, because I frequently find their ova in the excreta from such cases, and a course of thymol usually brings the temperature to normal and permits convalescence to proceed without interruption.

Diagnosis.—A microscopical examination of the faeces is the only manner among the South African natives that the diagnosis can be made with certainty. The blood count is useless as a means of differential diagnosis, as so many other parasitic diseases are prevalent in this country, which cause an increase in the number of eosinophiles. This is well demonstrated by the following table, which shows the various species of worms found at post-mortem examinations;—

Tribe.	No. examined.	Bilharzia worms in liver.	Ankylostome	Round worms.	Tape worm.
Mozambique ...	215	102	139	17	1
Quelimane ...	129	60	81	32	1
Nyassa ...	52	16	27	10	2
Brit. Cent. Africa	10	4	2	1	1
Beira boys ...	3	1	2	0	0
Angoni ...	14	2	3	1	0
Shangaan ...	65	23	32	27	2
Tonga	6	0	1	1	2
Myambaan ...	106	28	30	36	7
Mtyopi ...	40	9	15	8	8
Transvaal Basuto	12	1	1	0	1
Damara ...	19	0	2	1	0
Swazi	1	0	0	0	1
Pondo	6	0	0	1	3
M'Xosa ...	5	0	0	2	1
Mixed Races ...	9	0	0	0	0
Total ...	692	246	335	137	30

—G. A. Turner.—*Transvaal Medical Journal,* Sept., 1908. (J. A. W.)

Mine Tailings on the Rand (planting grass, etc.).—"I examined some samples of the tailings taken from such heaps some time ago. I therefore append an abstract of a report which I wrote in 1905 on behalf of the Municipality of Johannesburg. Four samples were received, which were described as follows:—1. From fresh tailings taken from the skip on the way to the dump. 2. From tailings about six month old, taken from just under the surface of the dump. 3. From tailings about twelve months old. 4. From tailings which had been exposed to the weather for at least six years. All samples were from the Ferreira dump. In No. 1 no appreciable amount of cyanide could be detected (the examination was not made until the sample was a few weeks old), but the presence of sulphides in considerable quantities was clearly evident. I am inclined to think, therefore, that the sterility of the dump is dependent more upon the presence of sulphides than of cyanides. The composition of the tailings is shown by the following figures:—Moisture, 5·47; loss on ignition, 1·10; silica, 90·76; iron oxide and alumina, 2·35; lime, 0·91; magnesia, 0·21; potash, 0·05; phosphorus pentoxide, 0·03; total, 100·87. The mineral ingredients necessary for plant growth—potash, lime, magnesia, and phosphoric acid—are thus present in sufficient quantities for the requirements of plants, the first named being perhaps most deficient. But the presence of oxidisable sulphide of iron renders the material sterile even if the absence of nitrogenous matter did not do so. The sulphide of iron undergoes slow oxidation, being slowly converted by absorption of oxygen into sulphate of iron and free sulphuric acid, which at first is neutralisd by the lime present Thus at first (No. 1) the tailings are distinctly alkaline in reaction, due to the presence of free lime, after some time (No. 3) they become neutral, while after some years (No. 4) they develop a distinctly acid reaction. The total sulphur present diminishes as time elapses, owing to the formation of soluble sulphates and their removal by rain. Determinations of the total sulphur and soluble (in water) sulphur gave the following:—Total sulphur; Fresh, 1·21; six months, 1·21; twelve months, 1·10; six years, 0·24%. Soluble sulphur: Fresh, 0·012; six months, 0·010; twelve months, 0·011; six years, 0·029%. It would thus appear that during the first year the oxidation and removal of the sulphur was slow, but that afterwards it proceeded more rapidly, but that even after six years there still remained some unoxidised sulphur. Of course, these deductions are based on the assumption that the material when deposited was of similar composition during the whole period.

In order to render the tailings more capable of supporting plant life, it seems to me to be necessary—1. To destroy the oxidisable sulphides—pyrites, etc. 2. To increase their porosity and friability. 3. To add the necessary nitrogenous organic matter. 4. To neutralise the acidity which develops upon the oxidation of the pyrites.

No. 1 could only be done on the large scale by the air, and for this to occur No. 2 would have to be effected. No. 2 could probably be accomplished by mixing the surface of the dump with granular 'clinker' from engine furnaces, rejecting the fine ashes and the very coarse clinker. This admixture would render the surface porous, and would favour oxidation by facilitating the access of oxygen. The oxidation would be favoured by periodically watering the surface. No. 3 would be best done by adding stable manure or similar material *after* Nos. 1 and 2 had been in operation for some time. If added too early, the organic matter being itself oxidisable would tend to prevent the oxidation of the pyrites. No. 4 could be best effected by the addition of powdered lime added simultaneously with the stable manure. As to quantities, it is difficult to advise; probably 10 or 12 tons of clinker per acre would suffice. The granular clinker should be spread on the surface and harrowed or raked in, and an occasional harrowing, say, once a month, would probably be helpful, especially after rain or watering.

The stable manure should be applied at the rate of 20 or 25 tons per acre, and powdered lime to the amount of, say, 2 tons per acre, well harrowed in, followed after a month or six weeks by the grass or other seeds.

I fear the above method would be expensive, but I cannot suggest any cheaper plan likely to be successful. Of course, another method, but I fear almost too laborious and expensive, would be to cover the dump with a layer of soil to a depth of 6 or 8 in., and plant the seeds in that.

Our agrostologist, Mr. Burtt-Davy, suggests Bermuda grass, 'Oprecht kweek gras' (*Cynodon Dactylon*) as a suitable grass for the purpose of trial.

Some relief from the annoyance of dust blowing from the dump heaps might possibly be derived from the use of shelter belts of trees planted round the heaps, but, of course, a considerable time would be required for the trees to grow sufficiently high to be effective. Our conservator of forests, Mr. Legat, advises *Eucalyptus viminalis* as the most suitable tree, to be planted in belts at least a chain wide around the dumps, the trees to be planted 6 ft. apart, and those nearest the dump to be allowed to be highest, those furthest away lowest, so that the wind would be turned upwards by the belt. I am afraid, however, that the problem is a difficult one, and in any case would entail considerable expense and labour. I regret that I cannot suggest any simpler and less troublesome plan."—Herbert Ingle. — *Transvaal Agricultural Journal*, July, 1908, p. 636. (G. H. S.)

Iron-ore Cement in Sea Water. — "A pamphlet has been published by a German cement company covering a series of tests of the resisting and enduring qualities of iron-ore cement and Portland cement, from which the following conclusion was reached by the company:—

Through our experiments we arrive at the conclusion that the resistance of the Portland cement which we used for comparative tests depended upon two points, namely the method of manufacture and the percentage of alumina. Those made under the wet process invariably showed greater constancy of value than those made under the dry process, supposing that the contents of alumina did not vary too much. The amount of alumina, without doubt, is the more decisive factor of the two. All experiments were convincing evidence of the fact that a cement disintegrates the sooner or suffers so much more serious damage within a given time the higher its contents of alumina. Portland cements, being low in alumina, show greater resistance in the presence of gypsum or in sea water than those of a high percentage of alumina, but, in the long run, even the former go to destruction. Iron-ore cement, as previously stated, passed the severest tests without the slightest damage its contents of alumina never exceeding 1·5%.

All the foregoing statements warrant the declaration that well-burned iron-ore cement is to be used for marine construction, for sea-water canals, and for all cases in which the concrete comes in contact with sulphuric acid salt solutions."—United States Consular Report.—*Mines and Minerals*, Aug., 1908. (A.R.)

Reinforced Concrete.—Its efficiency has been thoroughly demonstrated, and it is now recognised as a permanent method of construction; it is proof against the deteriorations to which other materials are subject. It is fire-proof, resists earthquakes, is rust-proof, practically everlasting, and its strength is continually increasing with age.

The constituents from which it is made can easily be obtained in almost any locality, *i.e.*, sand and crushed stone or ballast; only the steel and cement being brought any distance. In the early stages crushed stone concrete is a little stronger than ballast concrete, but at the end of 30 days ballast concrete is just as strong as stone concrete, and from then onwards its strength begins to surpass it.

It has recently been proved that a moderately fine sand will give a stronger concrete than a sharp coarse sand; sand taken from beds in contact with impure water kills the action of the cement and should be condemned. Pit or bank sand is a very satisfactory kind to use. Cement should be of the very best grade and be subjected to many mechanical tests during the progress of the work.

Concrete mixing is an important feature, only clean water free from acids and strong alkali should be used in mixing. The resultant should be that known as a wet mixture; no concrete which has once begun to set should be deposited.

Concreting should never be done in freezing weather; concrete should be reinforced in a vertical as well as a horizontal plane.

Reinforced concrete is the best form of construction when properly handled, and the worst when improperly handled.—M. Kahn.—*I. of Engineers and Ship Builders of Scotland*. (C.B.S.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

CYANIDE PROCESS. By E. B. WILSON, E.M. Fourth Edition, revised and enlarged. Price, 6s. 6d. (New York: John Wiley & Sons. London: Chapman & Hall, Limited.)

"Cyaniding is now perhaps the best known process in gold milling, yet since the first edition of this work was published in 1895, although many theories have been advanced and proved or found worthless, there remains still much to learn. Cyaniding has widened in scope to such an extent that ores are now successfully treated which were once considered beyond the capabilities of the process.

The present edition is designed especially to describe the improvements that have taken place in the treatment of slime since the last edition of the work published in 1902. No unnecessary details are incorporated in the work, and by thus dealing almost solely with practical facts about the process, rather than with mechanical details, the author has been able to publish this little work in a terse and lucid form.

The following apparatuses are described: Butter's distributor with Blaisdell improvements, Sweetland press, Moore's stationary and vacuum filter, Blaisdell pressure filter, Kelly pressure filter, the Adair-Usher slime process (introduced in South Africa in 1907), and the Nicol's slime settling process. All these processes are clearly illustrated by diagrams, and the rate of capacity of the various appliances connected with each process distinctly stated, together with their respective advantages.

Considerable space is devoted to the application of electricity in cyaniding, both time, cyanide, and zinc being saved by its adoption, as well as other advantages gained through its employment. This branch of the subject is thoroughly dealt with in the various chapters on electrodes, current, anodes, and cathodes.

A chapter is devoted to the Siemens-Halske and Pelatan-Clerici processes, another to general information on cyaniding, and an appendix at the end of the book gives instructions how to proceed with a treatment of a case of cyanide poisoning in the absence of medical assistance. The work is carefully and concisely written, and is thoroughly indexed." — *London Mining Journal*, Oct. 10, 1908. (W. A. C.)

HINTS ON AMALGAMATION AND THE GENERAL CARE OF GOLD MILLS. By W. J. ADAMS. San Francisco., Cal., 1908; *Mining and Scientific Press.* Pp. 120, illus. For sale by *The Mining World.* Price, $2.

"There are a number of books on gold mill practice, and while some have merit by reason of the experience of their authors, others fail to satisfy the popular demand, because they are written in the technical language which few practical millmen can understand. Mr. Adams, in revising the second edition of his excellent treatise, has taken pains to employ the language that can be understood by all, and wisely condensed his explanations of the important features of gold milling, so that the busy man need not lose time in gaining the knowledge which he desires. The title of the book has been well chosen, for the reason that 'hints' on amalgamation are wanted alike by the experienced and inexperienced millman."—*The Mining World*, Sept. 19, 1908, p. 449. (W. A. C.)

GENESIS OF METALLIC ORES AND OF THE ROCKS WHICH ENCLOSE THEM. By BRENTON SYMONS, M.Inst.C.E., C.M.E.; 154 illust. Price, 7s. 6d. With map, 8s. 6d. (London: *The Mining Journal.*)

"Of the geological formation and mining possibilities of many lands, Mr. Brenton Symons has written much that is valuable, lucid, and to the point. In the latest published of his works, 'The Genesis of Metallic Ores, and of the Rocks which enclose them,' he has endeavoured, and that with marked success, to bring the complex scientific problem of the genesis of ore deposits out only within the comprehension of the average man in the street, but to make it intensely interesting reading, and that much on the lines that inspired Camille Flammarion when he brought astronomy within the understanding of the average lay intellect. In his work, which is well printed, well written, and admirably illustrated, Mr. Symons, avoiding all scientific phraseology, which, however ordinary it may be to the student, is caviare to the multitude, leads the reader on step by step from the hypothesis of creation to the certitude of the created as exemplified in that portion of the earth's crust which he so lucidly classifies. The publication of the work should be particularly welcome in that it fills the hiatus between the primer, which is of little use outside the sixth standard, and the average text-book which assumes the student's knowledge to be on a par with that of the author."—*The Mining Journal*, Sept. 26, 1908, p. 406. (W. A. C.)

GOLD AND SILVER. Comprising an Economic History of Mining in the United States, the geographical and geological occurrence of the precious metals, with their mineralogical associations, history and description of methods of mining and extraction of values, and a detailed discussion of the production of Gold and Silver in the world and the United States. By WALTER R. CRANE, Ph.D. First Edition. Pp. 727. Price, 21s.(New York: John Wiley & Sons. London: Chapman & Hall, Limited.)

"Although the title on the cover of this work ("A Treatise on Gold and Silver") would indicate that it covers a world-wide range, it is practically, as shown in the preface, confined to the occurrence of the precious metals in the United States, and will ultimately form a part of the Economic History of the United States, to be published by the Carnegie Institution of Washington.

This compilation should prove a valuable work of reference, especially as the original sources are very fully given. About half of the work deals with the occurrence and geological distribution of the metals —over 100 pages with the mining, and over 200 pages with the treatment of the ore. There is also, in addition to the numerous valuable tables which appear with the letterpress, an appendix of tables dealing with the discoveries of districts, the occurrence and mineralogical association of gold and silver, and the outputs and values for individual districts and mines. A useful bibliography is also appended.

The work will, under any circumstances, be found of extreme value as a book for reference, and the sections dealing with the extraction of values, although, as already stated, often unduly brief, are ably dealt with, present the processes clearly and correctly, and are well up to date, and may be read with advantage before consulting other treatises which have been written on special branches of treatment."—*London Mining Journal*, Sept. 26, 1908, p. 405. (A. R.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 418/08. Louis Schedl. Improvements in jockey or rope grips for mechanical haulage systems and the like. 19.10.08.

(C.) 424/08. Heinrich Flottmann. Improvements in rock drilling machines. 23.10.08.

(C.) 425/08. Patrick Joseph Donovan. Improvements in filters. 22.10.08.

(C.) 426/08. Charles Budd Richards. Fluid operated tools. 23.10.08.

(C.) 427/08. Louis William Greve (1), Charles Budd Richards (2). Pneumatic hammer and drill bit. 23.10.08.

(P.) 428/08. Alexander Cullen Whitehead. Adjustable tube scraper. 26.8.08.

(P.) 429/08. Robert Clifford de Lorme (1), Wilfred George de Lorme. Improvements in tipping trucks. 26.10.08.

(C.) 430/08. William McVitty. Improvements in driving means for rotary ore feeders. 26.10.08.

(P.) 431/08. Henry James Shedlock Heather. Improvements in lightning arresters and other apparatus for protecting electric circuits against the effects of abnormal rises in voltage. 27.10.08.

(P.) 442/08. Wilhelm Mauss. Improvements in self-feeding percussive apparatus. 4.11.08.

(P.) 443/08. Hugh Henry Boyd Stewart. Improvements in the construction of dams, weirs, quay walls, retaining walls and the like. 5.11.08.

(P.) 445/08. James West. Improvements in grips for cable haulages. 5.11.08.

(C.) 447/08. Sir James Heath (1), William Hales Turner (2). Improvements in or relating to the drilling of rock or the like and apparatus for use in connection therewith. 6.11.08.

(C.) 449/08. Henry Rainhard. Improvements relating to the manufacture of metal tubes, rods, and the like. 6/11/08.

(P.) 450/08. Alexander John Arbuckle (1), Alfred Osborne (2). Improvements in means for separating pulverised ore or other comminuted solid matter from liquid. 7.11.08.

(C.) 452/08. Fritz Wagner. Improvements relating to a method of and means for obviating the vibration of main pressure valves. 13.11.08.

(C.) 453/08. Mortimer Frederick Meiville. Improvements in apparatus for the production of carburetted air. 13.11.08.

(P.) 454/08. George Hans Kirsch (1), Walford Robert Dowling (2). Improvements relating to tube mills and similar grinding apparatus. 13.11.08.

(C.) 455/08. Henry Wolfgang Foust. Ore concentration jig. 13.11.08.

(C.) 456/08. James Palemer Hearn. An improved pendulum pump worked with an adjustable weight. 18.11.08.

(P.) 457/08. Hermann Stadler. Ventilating structural sheet. 18.11.08.

(C.) 459/08. Sidney Smith. Improvements in carburetting apparatus. 20.11.08.

(P.) 460/08. Tullic Bubola. Improvements in or relating to tanks or vats for the treatment of crushed ore products. 21.11.08.

(C.) 461/08. Harry Murson Thomas (1), John Wylie Craig Niven (2), John Greig (3), Robert Niven (4). Improvements in furnace for heating hand rock drills, machine rock drill bits and similar edge-tools preparatory to the sharpening of same. 21.11.08.

(C.) 462/08. Robert Findlay Sturrock. Improvements in furnaces for boilers, 23.11.08.

(P.) 463/08. Richard Colson. An improvement in dropper and posts and similar articles used for fencing and similar purposes. 23.11.08.

(P.) 464/08. Thomas Passe. Improvements in means for imparting the intermittent rotary motion to the pistons of rock drilling machines. 23.11.08.

(P.) 465/08. Charlton Frederic Bayly. Device for regulating the underflow of cones, spitzkasten and the like. 24.11.08.

(C.) 468/08. Hubert Elwell Smith. Improvements in and relating to carburetters. 27.11.08.

(C.) 469/08. Richard Scherl (1), Emil Falcke (2), Paul Froehlich (3), Alfred Kuerth (4). Improvements in or relating to gyroscopic mechanism. 27.11.08.

(C.) 470/08. Heinrich Flottmann. Improvements in packing for rock drills. 27.11.08.

(C.) 471/08. Heinrich Flottmann. Improvements in rock drills and other pneumatically operated tools. 27.11.08.

Changes of Addresses.

Members and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

BAWDEN, F. A., *l/o* Germiston ; Ingleside, Redruth, Cornwall.
BELL, H. C. F., *l/o* Cleveland ; Cala, Tembuland, C.C.
BELL, S. W., *l/o* Rhodesia ; Tankerville House, Newcastle-on-Tyne, England.
BRICKHILL, H. G., *l/o* Germiston ; Nigel Deep, Ltd., P. O. Box 50, Nigel.
CARRUTHERS, J. T., *l/o* Fordsburg ; 670, High St., Brixton.
CHAMBERS, A. L., *l/o* Johannesburg ; Subeni Coppers (Natal), Ltd., Subeni P. O. via Vryheid.
DRAKE, FRANCIS, *l/o* New York ; 310, Kohl Building, San Francisco, Cal., U.S.A.
HERBERT, S. A., *to* Simmer Deep, Ltd., P. O. Box 178, Germiston.
INGLIS, A. B., *l/o* Cleveland ; Thornton Little G. M. Co., Barberton.
OGLE, F. B., *to* Simmer Deep, Ltd., P. O. Box 178, Germiston.
PRICE, W. S. V., *l/o* Johannesburg ; Ginsberg's Cottages, Benoni.
SIMMONS, T. W., *l/o* Johannesburg ; Consolidated Langlaagte Mines, P. O. Box 15, Langlaagte.
TURNER, H. F. S., *l/o* Johannesburg ; New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston.
URQUHART, W. G., *l/o* Krugersdorp ; Simmer Deep, Ltd., P. O. Box 178, Germiston.
WESTON, E. M., *l/o* Brakpan ; 19, Jumper St., Jeppes Extension, Johannesburg.
WOODBURN, J. A., *l/o* Johannesburg ; Messina Development Co., Ltd., Grenfell Camp, via Pietersburg.

THE JOURNAL

OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. IX. DECEMBER, 1908. NO. 6.

Proceedings AT Ordinary General Meeting, December 19, 1908.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday evening, December 19th, Mr. R. G. Bevington (President), in the chair. There were also present:—

29 Members: Dr. J. Moir, Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, C. B. Kingston, A. Richardson, G. O. Smart, H. A. White, Prof. J. A. Wilkinson, A. Avent, S. Beaton, W. A. Caldecott, F. T. Chapman, G. Goodwin, J. H. Harris, G. A. Lawson, J. Lea, C. D. Leslie, H. S. Martin, F. T. Mumford, F. B. Ogle, O. D. Ross, C. E. Rusden, H. Scarf, A. Thomas, H. M. Thomas, O. Tonnesen, J. F. Walker.

5 Associates: Messrs. D. Dowling, N. N. Newland, W. A. C. Tayler, W. E. Thorpe, J. Whitehouse.

10 Visitors and Fred. Rowland, Secretary.

The President: I beg to move that the minutes of the last ordinary monthly general meeting, as printed in the November *Journal*, be confirmed.

This was seconded and agreed to.

NEW MEMBERS.

Messrs. H. A. White and E. H. Croghan were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all candidates for membership had been duly elected, as follows:—

DARROW, WILTON ELMORE, Sutter Creek, Amador County, California, U.S.A. Gold Metallurgist.

WYNDHAM, HERBERT HENRY, Aurora West United G. M. Co. Ltd., P. O. Box 26, Maraisburg. Cyanide Manager.

The Secretary: Since the last meeting of the Society the following associates have been admitted:—

THORBURN, JOHN MACKAY, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Analytical Chemist.

TURNER, HUBERT FREDERICK SIDNEY, New Primrose G. M. Co., Ltd., P. O. Box 193, Germiston. Sampler. (*Transfer from Student Roll.*)

GENERAL BUSINESS.

The President: Since our last meeting the Society has paid a visit to the Premier Diamond mine. I was very pleased to see that we had a very large attendance, and I am quite sure that all those members present were pleased with their visit, as almost every one who went out expressed their satisfaction to me and their great interest in the works. As an old resident of the Transvaal and of South Africa, I may say that many years ago I visited Kimberley and spent some time around the diamond mines there. From that time until our visit to the Premier mine I have had no connection with diamond mining, and it greatly impressed me to see the advance that has been made. For instance, in the days I speak of, in Kimberley there was no thought of crushing diamond ground. Such a thing was against all ideas of procedure in regard to diamond mining, because it would have broken up the diamonds. Now we find that the Premier Diamond Company does not spread its blue ground out, as the old Kimberley companies did, over numbers of square miles of ground, but takes it right out of the mine and puts it first through a gyratory crusher, then through rolls and then it goes on through the washing and diamond selecting processes. This method of procedure was unheard of in the old Kimberley days, and as far as the Premier Diamond Company is concerned, I do not think they care about big diamonds. They have had one Cullinan diamond and they do not want any more. That diamond has become the diamond of the world. Their main reason would be, of course, the fact that large diamonds are not easily saleable, while medium sized and small diamonds are, and they may as well be crushed by this crushing process as split by the cutting process afterwards. Of course, the output has very much increased through the winning of diamonds from the diamondiferous ground almost immediately it is taken out of the mine. I am sure we were all delighted with our visit, and I hope that on some future occasion we shall have an opportunity of going there again. The welcome we met with and the kind manner in which we were shown round the whole place could not have been exceeded, and I should like to propose on behalf of the Society a hearty vote of thanks to the directorate and management of the Premier Diamond Company for the way in which they received us, for showing us everything possible, and giving us all the information we asked for.

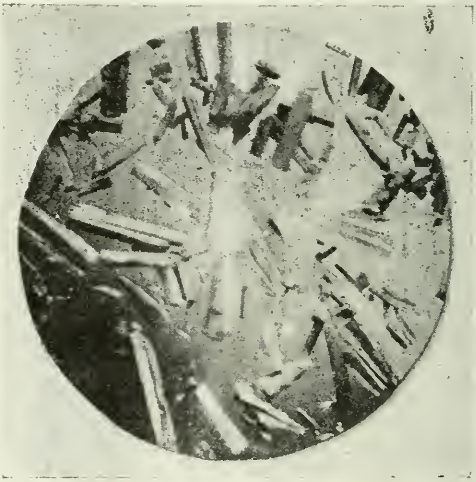
There is one other matter I should like to mention under the head of general business. In the October *Journal* you will notice a table which has been compiled by our Past-President, Mr. W. A. Caldecott, on the vat capacity of circular tanks. It is a very valuable table, and the thanks of the Society are due to Mr. Caldecott for having placed it in our hands. It should be of great use to cyanide managers and engineers on these fields who have to deal with circular vats and tanks. I may also mention for the benefit of any cyanide manager or members of the Society who may wish to have separate copies, that they may be obtained on application to our Secretary. I should like personally, and on behalf of the Society, to thank Mr. Caldecott for the trouble he has taken in compiling this table and giving it to the Society.

GOLD CRYSTALS.

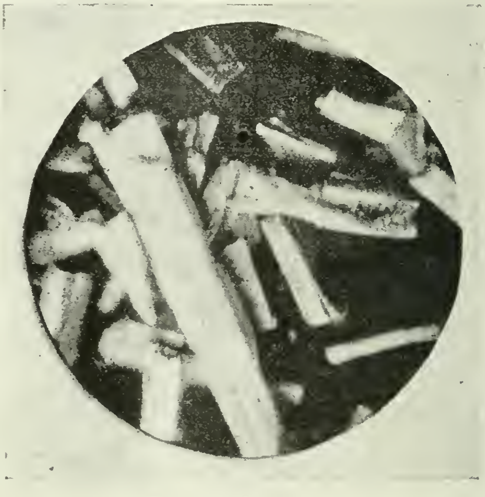
Mr. A. McA. Johnston (*Vice-President*) *:* For a few minutes to-night I would ask your indulgence in bringing to your notice some crystals of gold which Mr. A. Mennie let me have that I might examine under the microscope. As most of you are aware, when the particles of free gold come into contact with the clean mercury surface they are, to use a milling term, caught. The process, however, involves a physical action rather than a chemical one, that is to say, the gold does not dissolve in the mercury. To obtain a solution of the finest gold prills in mercury is rather a lengthy process, indeed the solubility of gold in mercury is very small indeed.* There is, however, a point when, under certain conditions, a solid solution of mercury in gold is obtained, one important condition being the element of time.

This solid phase has been obtained in these crystals, though as I will point out, the crystals are in themselves not quite perfect gold mercury crystals, inasmuch as some at least of the mercury has been dissolved out. Mr. Mennie thus describes their origin. "These gold crystals were obtained from mercury which had been used in the Molloy electrical process many years ago. There were about 2 tons of mercury which had been lying there from one to two years after retorting. Thinking there might be some gold left therein, I started straining it with the result that 2 oz. or so of gold were recovered, and much of the amalgam was distinctly crystalline. A small part of this I treated with nitric acid, and these crystals represent the product. Before treatment with nitric acid, some of the crystals of amalgam were more than ¼ in. long, but after treatment much of the gold broke up."

I have here for your inspection some photographs showing two magnifications of these crystals. As you will see, very few of them are really perfect in shape, though the general tendency is to form elongated hexagonal figures. A close inspection shows that numerous little



Magnified 9 Diameters.



Magnified 28 Diameters.

---

* See Louis—"Handbook of Gold Milling,"

holes exist from which the mercury has been eaten out by the acid with which Mr. Mennie treated the original crystals. With regard to appearance, the crystals are dull yellow in colour with sharp, ragged edges. The irregular faces of most of the crystals are mainly due to the action of the acid, though there are also some whose double appearance is from the superimposing of one crystal on another. You will have noticed that I speak of these crystals as having existed as a solid solution of mercury and gold. In doing so, of course, I take the more prominent view, as there exists the possibility that these metals form not this but an intermetallic compound from which the nitric acid has dissolved the mercury.

The President: Our thanks are due to Mr. McArthur Johnston for bringing this interesting little note before us. With regard to this matter I would like to state, and I dare say others have noticed the same thing, that in cleaning out the mercury traps, particularly in deep mercury traps you will find a very large proportion of crystalline matter, especially when gathered from the bottom of the traps. That, I think, shows that this crystallisation of mercury and gold occurs very much more rapidly than Mr. Louis thinks. I have noticed it pretty well every month when the mercury traps are being cleaned out. I have not gone to the length which Mr. Mennie has, of treating this crystalline matter, but I have not the slightest doubt it could be done. I do not know whether any other mill manager or member has noticed the same thing.

Mr. A. McArthur Johnston (*Vice-President*): I very much doubt whether what the President is talking about is the exact thing I mean; the solubility of gold in amalgam is exceedingly small.

The President: I shall make it my business to get some of this crystalline amalgam and treat it with nitric acid and see if we can get the same results.

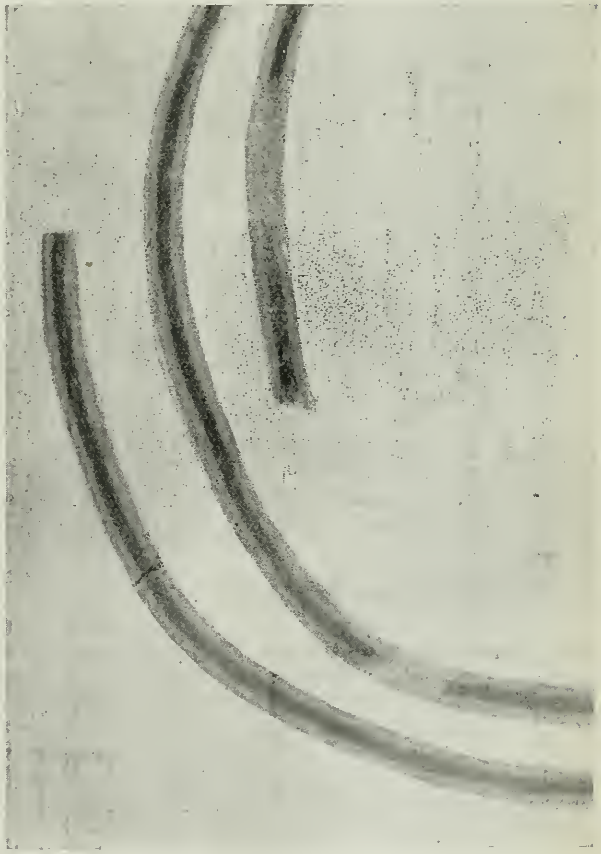
Mr. G. O. Smart (*Member of Council*): I have noticed this crystalline amalgam in mercury traps and other parts of the mill. I think you will find it merely pieces of copper from the detonators.

The President: I hardly think so, for they appear to me to be of cubical crystalline form, but I certainly will try it. I quite agree that the film of gold may be so thin that it will crumble, and one may not be able to obtain complete crystals.

The Secretary: Mr. Thomas, the Inspector of Explosives, has asked me to read a short note from him to-night, as follows:—

TESTING OF SAFETY FUSE BY X RAYS.

Mr. Jas. Thomas (*Member*): I wish to bring before the Society an item which may be of interest to some of the members. About four years ago, during the course of some investigations, I tested the method of looking for defects in safety fuse by means of the X Rays, and I had several coils of safety fuse examined by these means, but without discovering any breaks in the powder. Recently, however, having received some complaints in respect to some fuse, I had samples of it submitted to an X Ray examination, with the result that two spaces were found in the powder column. I have pleasure in submitting a photograph illustrating these breaks. This experiment will, I think, prove that it is possible in cases of mis-fires to ascertain whether the column of powder is continuous. I am hoping to make further experiments in this direction, the results of which I may be able to communicate at a later date.



X Ray Photograph of Safety Fuse showing Breaks in Powder Column.

The President: All these matters, whether short contributions or otherwise, are of great interest to us, and we must thank Mr. Thomas for having brought the results of his experience before the Society, and express the hope that we may have his further notes on the subject.

Mr A. McArthur Johnston (*Vice-President*) then read the following paper :—

ROUTINE ASSAYING ON A WESTRALIAN MINE.

By W. B. Blyth (Associate).

It is now some years since we were favoured with that interesting and instructive paper, "Routine Assaying on the Rand," read by Mr. A. Whitby.

It may be interesting to Rand assayers and to members of the Society generally to have a description of some present day Westralian practice ; and my object will be gained if this paper rouses that interest to a spirit of healthy criticism. It may be as well to explain here that the following notes attempt to describe in detail the routine assaying work of one of Westralia's 100 head free milling propositions. The ore is a clean blue quartz, and the country rock is diorite and the plant crushes about 22,500 short tons per month. The practice described, though differing in minor details from other offices, is probably representative of best Westralian practice.

Sampling.—Reduction plant samples, consisting of mill heads, sands, concentrates, as well as tailings, heads and residues, treatment and solution samples are taken by shiftmen, collected by assay boy, and delivered at the assay office at 8.30 a.m.

Mine Samples.—Consisting of truck, stope and development samples marked with consecutive numbers, are taken by underground samplers, and vary in weight from a few lbs. to 50 or more lbs. of any sized rock. Development samples are distinguished by having a D on the label in addition to the usual number. This is done to insure that additional precautions be taken in cleaning the crushing apparatus, and also that these samples be assayed in duplicate. Mine samples are just put through a Gates' gyratory power driven sample crusher, delivering a $\frac{1}{4}$ in. product. This is spread evenly over an iron plate, and a representative portion taken for grinding with a wide thin spatula. A duplicate sample is taken here, and one is kept as a check for a few days. The first sample is reduced down to a product which will all pass through a 60-mesh screen in an ordinary gyratory sample grinder. The machine is cleaned after each sample with a jet of compressed air. In the case of development samples, the machine is pulled to pieces after each sample is ground, and each part is subjected to the air jet. This is done to avoid any possibility of salting. After grinding, the sample is placed in a numbered paper bag, and is ready for assay.

For reducing the plant samples, the ancient bucking board device is used, one being kept exclusively for concentrates, and any known rich samples, and the other for residues and sand samples. The samples are mixed and quartered down to the required size while wet, and dried on an iron plate over a wood fire, care being taken to avoid roasting. Sands, slimes, etc., are made to pass through a 60-mesh screen. Concentrates and rich samples through a 100. The samples are ground in order of their respective richness to avoid salting by dusting, etc.; the lowest in value being ground first. As ground, the samples are placed in labelled paper bags, and are ready for assay. A small vacuum filter pump is used for washing treatment samples containing cyanide gold solution.

Furnaces.—The furnace room is 18 ft. x 18 ft. with a concrete floor. There are two rectangular coke fired wind furnaces, 17 in. x 17 in., taking charges of 25 two-ton or 20 four-ton assays respectively. The muffle furnace is attached to the side of the main stack, and was designed by the assayer. The excessive heat at the base of the stack and directly above the muffle, which in the Battersea type of furnace is wasted, is here used to heat a second muffle which is placed directly above the first. The furnace is charcoal fired, and each muffle, "Battersea size J" holds 25 cupels.

Weighing and Fluxing.—A stock flux is prepared consisting of soda, 10 parts ; borax, 4 parts ; litharge, 3 parts. Battersea Fluxing pots Nos. 10 and 11 are used. These average 6 fusions per pot, and owing to their shape are found to dust less than the ordinary Battersea G type. The sample plus flux is ground in a mortar, tipped on to a piece of glazed sampling paper, and transferred to the crucible, to which sufficient silver for parting has already been added. The crucibles are placed in rotation in a wooden partitioned tray, each charge is covered with a thin layer of borax, and the tray is removed to the furnace ready for charging. The fire is levelled down and covered with a layer of cold fine coke. No packing between the crucibles is allowed, as it is found that packing causes the bottom to get heated first, and this causes dusting. The weights of ore taken for assay are respectively as follows :—For concentrates and any known rich samples 1 A.T. ; mine samples, mill heads, concentrate residues, etc., 2 A.T.; sand heads and residues, slime heads and residues, and treatment samples, 4 A.T.; while for cyanide solution 100 c.c. are taken. For two A.T. of ore, $5\frac{1}{2}$ A.T. of flux are used; for 4 A.T. 10 tons of flux and so on. The stock flux is used for everything, including the assay of concentrates, cyanide slags, etc. In the case of raw concentrates (containing about 16% of S) no reducing agent is employed, and two nails are introduced into each pot. The slag produced is fluid and entirely free from gold. Under ordinary circumstances a charge will take from 30 to 40 minutes to fuse properly. The pots are then poured, buttons detached, cleaned and cupelled in the usual way. Fluxing without any borax has been tried, and as the ore consists of nearly clear quartz, the results were satisfactory. It was found, however, that the presence of a little borax tended to make the slag more fluid, and to shorten the time of fusion. For assaying solutions the old evaporation method has been discarded in favour of the zinc lead acetate method. Briefly described the process is as follows:—100 c.c. of solution are placed in a beaker with approximately 2 gms. of Zn shavings, and 15 c.c. of a 10% solution of lead acetate are added. The beaker is placed on an iron plate over a Primus stove, and the contents brought to the boil. 20 c.c. of strong commercial HCl are then added and left till all the superfluous Zn is dissolved. The liquid is now decanted off, holding back the spongy lead ppt. with a glass rod. The precipitate is then transferred to a piece of filter paper, and the moisture removed by pressing between the finger and thumb. The lead pellet together with enough silver for parting is wrapped in a piece of lead foil and cupelled. In the case of concentrates, and very foul solutions, the lead will be found to break up and the solution will have to be filtered, the filter paper being wrapped in lead foil and the whole cupelled. The accuracy of the assay will not, however, be impaired. This method has many advantages over the evaporation assay. The whole process may be completed, and the results out in an hour if necessary. Results of comparative tests show it to be quite reliable.

Parting and Weighing.—After cupellation the beads are flattened and placed in a row. A number of 000 Berlin porcelain crucibles are half filled with parting acid (2 parts distilled H_2O to 1 part pure HNO_3), placed on a piece of asbestos millboard, over a Primus stove and heated. The beads are now dropped in and parting proceeds immediately. When finished the board and contents are removed to cool. The acid is decanted off down a glass rod, and the gold washed once with distilled water. The washing apparatus consists of a glass jar elevated a few feet; to the tap of which is attached a piece of $\frac{1}{4}$ in. rubber tubing. The rubber tubing terminates in a piece of bent glass tube drawn to a fine point. The pressure is regulated by keeping the rubber tubing between the finger and thumb. The bent glass serves first for decanting the acid off, and then for filling the crucible with water, and decanting off again. The crucibles are placed over the Primus to dry, and are finally annealed round the top of the Primus, at a red heat. One man can part about 75 assays in 45 minutes. All these operations are carried out in rotation; each separate tray of 25 assays being kept intact right up to the time of weighing. Any possible mixing is therefore confined to the one fire. The rule is to work from left to right, and as this movement becomes almost mechanical, cases of mixing are practically unknown. For weighing the gold prills two balances are in use. For mine concentrates and samples, where the unit of value is taken at 4s. an Oertling No 9 balance is employed. For residue and treatment samples where the utmost accuracy is necessary, an Ainsworth No. O balance is used. This balance has a 5 in. beam, and is sensitive to $\frac{1}{13000}$ gr. A quick method of obtaining the required oscillation, is to have a second rider attached to the left hand rider arm. The first rider is placed on the estimated mark; and the second rider lowered gently, close to the Zero mark on the other side, just as the pan supports are removed. The result is an immediate oscillation in the right direction. With practice the required distance of oscillation is obtained every time. As the values are now reported in shillings, the assay ton was altered for the sake of convenience to 309·895 grs. Taking 2 A.T. of this weight a gold bead obtained weighing ·001 grs. indicates a value of exactly 4s. per 2,000 lb. ton instead of 4·25s. or 1 dwt. as with the old A.T. Taking 4 A.T. ·100 gr. shows 2s. per ton. With the Ainsworth balance and a 4 A.T. charge, the values are reported to the nearest 6d. as the balance is sensitive to this limit. From 100 c c. of solution ·001 grain gold = 1s. 6d. per ton. This is a convenient measure to take. Grain weights are employed in this office not from any prejudice in their favour, but there has been a stock on hand for some years. The ordinary mine samples are assayed singly, all others including developments are done in duplicate.

Bullion Assays.—The sample is taken with a drill from each end, and the middle of the bar, and is received in the Assay office in the shape of borings. After cleaning well with a magnet, and blowing out any adhering dust, etc., ·5 gr. is weighed out in triplicate. Silver wire in the proportion of 2·5 of silver to 1 of fine gold, is added to two of the samples which are then wrapped up in pellets with about 9 grs. of lead foil. The third sample is cupelled for the base with lead alone. Morganite No. 7 cupels are used for this work, and the cupels are left in the muffle for a few minutes after the finishing point, as indicated by the display of colours. The third assay may be removed, cleaned and weighed right away; the difference in weight now and prior to cupellation represents the base metal contents. The others are cooled slowly in front of the muffle to avoid spitting; then cleaned, hammered, annealed and rolled into strips about $\frac{1}{4}$ in. wide, and the thickness of a visiting card. This is annealed again, and rolled into a cornet, and is ready for parting. For parting, the ordinary South African apparatus, consisting of a nest of 9 test tubes with perforated bottoms is used. The procedure is as follows: the first acid (HNO_3 sp. gr. 1·2) = HNO_3 200 c.c. + 250 c.c. H_2O is brought up to about 90° C, and the cornets introduced. They are boiled for 10 minutes after the red fumes cease, and are introduced into the second acid which is now boiling. The second acid is HNO_3 sp. gr. 1·3 = (324 acid + 124 H_2O). They are boiled in this for 15 minutes, lifted out, dipped in a bath of distilled water, and introduced into a solution of ammonia water, which dissolves off any adhering silver chloride. They are now tipped into porcelain crucibles, and the moisture removed with strips of filter paper, dried over a Primus stove, and annealed at a red heat round the top of another Primus. Oertling gold bullion weights are employed for weighing and the results of duplicates usually compare within ·02%. No allowance is made for surcharge or volatilization, as it is believed that these errors about counterbalance one another. *Cupels* are made in the office by a hand-fed machine, which can turn out 500 cupels per hour. The machine was designed and made on the mine. The cupels are stacked in a cupboard near the furnaces, about a week's supply being kept in advance, although, if necessary, they can safely be used within 24 hours.

The labour employed consists of an assayer and assistant, and two labourers working eight hours. About 4,000 assays are put through per month, or an average of 150 per working day. The office is closed on Sundays, one man coming in for half a shift to dry and prepare the treatment samples for next day. The cost per assay, on a 4,000 assay basis, is about 10·7d., made up as follows:—

	Total per Month.			Per Assay.
	£	s.	d.	d.
Labour ...	83	0	0	4·980
Supplies ...	86	9	5	5 188
Repairs ...	7	0	0	·445
Power ...	3	2	1	·185
Total	179	11	6	10·799

The cost of the principal stores consumed in the office is about as follows, delivered on the mine:—

	£	s.	d.	
Coke	5	7	6	per ton.
Charcoal	4	10	5	„
Soda Carbonate... ...	0	18	0	per cwt.
Borax	2	13	8	„
Litharge	2	11	3	„
Battersea No. 10, Pots	0	0	6	each.
„ 11 „ ...	0	0	8·7	„
Mabor	2	0	0	per cwt.
Nitric Acid	0	1	9	per lb.
Hydrochloric Acid ..	0	1	3	„

General Remarks.—A quick man can weigh and mix a fire of 25 assays, as described, in 25 minutes. Two hundred assays are generally put through on Mondays, owing to accumulations from Sunday. A number of outside assays are made for the public, and a substantial charge is made for same. A daily test is made of the lime used on the plant to determine the amount of available CaO. The sugar method is employed. This method only gives the soluble CaO, which is all that is of value to the metallurgist. Blank assays are run through to test each fresh lot of flux; but it is rare that any gold is discovered. The rooms comprise a weighing room, laboratory, furnace room, grinding room, while the crusher room is some distance away in another building. Coke bins, etc., deliver to the furnace room. A good many analyses are run through by the assayer, as well as a lot of experimental work. Standard solutions are prepared for the cyanide manager. The silver for parting is obtained in the form of No. 26 gauge wire. All the waste $AgNO_3$ from parting is kept, and the Ag precipitated with salt. This is periodically boiled with aqua regia, and run down for use again. Very little fresh Ag is required. Two fires of mine samples are prepared to go into the fire at 8 a.m., one boy coming in at 7.30 to light the fires. All the results are out by 3 p.m., and then preparations are made for next day.

The President: This is a paper which has been given us by one of our associates in a distant part of the world. I am sure you will agree with me in giving him a hearty vote of thanks for having contributed the paper. There is a good deal of information in it. I wish he had given us a sketch and particulars of the apparatus mentioned—a machine which can turn out 500 cupels per hour. I think it would be useful to many of our assayers. With regard to the various costs and the number of assays I think that can be best left to our assayer members to criticise.

Mr. A. McArthur Johnston (*Vice-President*): I should like to make one or two remarks to set the criticism going. The author says, "mine samples are just put through a Gates' gyratory power driven sample crusher delivering a ¼ in. product. This is spread evenly over an iron plate, and a representative portion taken for grinding with a wide thin spatula. A duplicate sample is taken here, and one is kept as a check for a few days." This method of keeping duplicate samples scarcely commends itself to me, as the product is too coarse to guarantee even an approximate result. I would recommend rather the grinding of the sample until there is a danger of the coarsest particles of gold being caught on the sieve. In regard to bullion assays the author makes this remark, "no allowance is made for surcharge or volatilisation, as it is believed that these errors about counterbalance one another." I am afraid I cannot altogether agree with that. There are many things which come into actual practice which would not allow Rand assayers to follow this practice. I know assayers who always have a surcharge, whilst others invariably have the opposite.

The President: Discussion has been commenced on this paper, and I trust you will follow it up. I notice that under the head of weighing and fluxing there is a mention of borax. I think it is very possible that our friend, Mr. White, may have something to say on this subject later on.

Mr. H. A. White (*Member of Council*): I would like to call the attention of members to the list of prices quoted at the end of this useful paper. It will be observed that borax costs about three times as much as carbonate of soda. It will also be noticed that the cost of assays is worked out to the thousandth of a penny. Again the author mentions that he has tried the use of a flux without borax, which he found quite satisfactory. I merely wish to ask members if they can see any inconsistency?

Mr. C. B. Kingston (*Member of Council*): Having been connected as consulting engineer and manager with West Australian mining for a number of years, this paper has a special interest for me. The writer deals with the treatment of the "mine samples" after they are taken, but there has been a good deal of interest attached to the taking of the samples in some of the West Australian mines, owing to the variety of deposits exploited, and the peculiar distribution of the values in some of the ores. In the upper zones of the principal mines about Kalgoorlie, for example, a great deal of very high grade telluride ore was found. Much of this was not treated on the spot, but was shipped to the smelters. At first there were serious differences between the shippers' and the smelters' samples, calling for a special investigation. Each party suspected the other in the orthodox manner, but at the same time reviewed their own procedure. It was at once found by the mining men that the ordinary method of mine sampling was unreliable for these ores. Trials were made of series of samples taken from the same ore in the stope face before blasting, after blasting, when filling into the shoots, when loading into the trucks, and again when dumping at surface. Differences of 25% in ores carrying 8 and 10 oz. were of common occurrence, and none of the results represented the real value of the rock, which was ultimately obtained by the regulation method of successively crushing, mixing and cutting down the whole of the broken ore in a series of steps until a truly representative sample was secured. The extreme difficulty of obtaining accurate results by the ordinary method of mine sampling in the case of the rich but irregular bodies of telluride ore in the upper zones of these mines, made the estimation of the ore reserves an anxious matter for the engineers and managers.

Prof. J. A. Wilkinson (*Member of Council*), read the following paper :—

GRADING ASSAYS AND GRINDING EFFICIENCIES.

By Arthur Yates, A.I.M.M. (Corresponding Member of Council.)

Since Messrs. Pearce and Caldecott read their excellent paper on the "Computation of the Crushing Efficiency of Tube-Mills" before the Society in September, 1906*, I have looked forward to the publication of further information on this interesting subject. However, no results have been placed before the Society showing the application of the proposed method of computation in practice.

With a view to stimulating further discussion on the value of grading analyses in connection with estimating the crushing efficiency and relative merits of machines, and also as a method for checking extraction results, I bring forward this paper.

We were just starting tube mills at Redjang Lebong, Sumatra, when Messrs. Pearce and Caldecott's paper was received, and I found it quite a boon in helping me to compare the work done here with the published results of tube milling elsewhere, as sizes of machines, horse powers, tonnages and gradings read separately, made a rapid and fair comparison of the work performed almost impossible.

---

* See this *Journal*, vol. vii., pp. 72—74.

At first my regular sample gradings were + 60, + 120, and − 120; as we crushed finer I changed to + 90, + 160 and − 160, and later to + 100, + 150, + 200 and − 200. Commencing in July, this year, I further subdivided the material to + 100, + 150 and + 200 sand, − 200 and elutriated slimes. Every morning the sands plant shiftman receives an average sample, for the 24 hours, of the pulp going to the tube mill amalgamation house; this he grades on 16 in. diameter screens (Greening's S.H.G. brass wire screening), grading wet for 20 minutes on each screen, drying and weighing the products, and then grading them dry for 10 minutes each and again weighing. The grading results are then calculated to percentages and entered in the log book. There is a marked difference between the wet and dry gradings, but it is fairly regular in amount, and by working the two methods one serves as a check on the other; only the final or dry grading is used in the calculations. The average difference between the wet and dry gradings for the 30 days of June, 1908, is here shown :—

	+ 100	+ 150	+ 200	− 200
Wet	5·05	19·09	9·55	66·29 = 100·00%
Dry	3·17	16·60	10·39	69·83 = 100·00%

The samples average 30 oz. dry weight. Every Friday morning a sample from the battery screen of Thursday's milling is graded; this is screened in the same way as the daily tube mill sample, and Thursday's battery and Thursday's tube mill samples are then calculated out for percentage efficiency.

SAMPLES FOR JULY 20th, 1908.

Battery. 70 stamps. 16 mesh screen.

	Wet.	Dry.	Efficiency Numbers.
+ 100	45·37	43·02	21·5
+ 150	8·94	7·82	16·6
+ 200	4·37	5·54	16·1
− 200	41·32	43·62	218·0
	100·00	100·00	272·2

Tube Mills. 4.

	Wet.	Dry.	Efficiency Numbers.
+ 100	1·84	0·92	0·5
+ 150	14·92	10·53	22·3
+ 200	8·85	10·89	32·0
− 200	74·39	77·66	388·5
	100·00	100·00	443·3

= 62·8% efficiency.

A weekly grading and assays are made on an average sample of all sands residues from Sunday to Saturday: the percentages of the different grades are calculated in the correct proportion to the total pulp produced, including the slimes made during the same period, and the results so obtained are checked against the corresponding week's tube mill samples and the battery screen sample of the Thursday of the same week.

WEEK ENDING JULY 4, 1908.

	Battery, 70 Stamps, 16 mesh.	Tube Mills, 3.	Cyanide, Sands and Slimes.
+ 100	46·6	4·4	3·1
+ 150	9·3	23·2	17·3
+ 200	4·3	8·7	16·0
− 200	39·8	63·6	63·6
	259·4	395·0	403·3
efficiency =		52·2%	55·4%

The elutriated slime is obtained by stirring the sample in a bucket full of water, allowing settlement to take place for 150 seconds and then drawing plugs (3) in the side of the bucket; the bucket is 11 in. high, and the lowest plug is $2\frac{1}{2}$ in. from the bottom of the bucket: the latter is filled and emptied five times. From Prof. R. H. Richards' tables (see Feb. *Journal*, 1907, p. 267) one finds that a particle of quartz of average diameter 0·05 mm. (= 0·002 in.) settles 1·78 mm. per second, therefore it will settle 267 mm. (= $10\frac{1}{2}$ in.) in 150 seconds.

For calculation of the efficiency numbers see this *Journal* p. 72, Sept., 1906. The work done in crushing is proportional to the reduction in diameter, per Rittinger's law, and, as for given weights of ore, the total surface area of all the particles in each lot varies inversely as their average diameters the relative surface exposed by each grade may be obtained by dividing its weight by the average diameter of particle in that grade.

Example.—

16		100		150		200		—200 screens
·0345		·0056		·0038		·003		·001 in. apert. of screens
	0·0200		0·0047		0·0034		0·002	mean diam. of particles between each two grades.

Battery Sample—16 Mesh Screen.

screens.	% dry weight.		efficiency number or relative surface
+100	43.02	$\frac{43\cdot02}{\cdot0210\times100}=$	21·5
+150	7·82	$\frac{7\cdot82}{\cdot0047\times100}=$	16·6
+200	5·54	$\frac{5\cdot54}{\cdot0034\times100}=$	16·1
−200	43·62	$\frac{43\cdot62}{\cdot002\times100}=$	218·0
	100·00		272·2

Battery sample = 272·2. Tube mill sample = 443·3. Therefore the additional relative surface exposed = 443·3 − 272·2 = 171·1 = 62·8% efficiency.

WEEKLY GRADING SAMPLE OF SANDS RESIDUES. Week ending July 4, 1908.

+100.			+150.			+200.			Sands −200.			Elutriated Slimes.			Assay General Samples.	
Per Cent.	Gold. dwt.	Silver dwt.	Per Cent.	Gold. dwt.	Silver dwt.	Per Cent.	Gold. dwt.	Silver dwt.	Per Cent.	Gold. dwt.	Silver dwt.	Per Cent.	Gold. dwt.	Silver dwt.	Gold. dwt.	Silver dwt.
6·2	6·2	52·8	34·7	4·6	44·9	12·7	3·2	36·3	33·3	2·4	30·1	13·0	1·0	19·0	3·3	35·9

Other gradings carried out here are, an average monthly sands charge sample and a sands residue sample, and an average fortnightly slimes charge sample and slimes residue sample.

An examination of the two grading analyses (see p. 190) shows clearly that for the Redjang Lebong ore with 9 days sands treatment and 17 hours slimes mixing, fine sands yield a better extraction at the sands plant than at the slimes plant, and as the capital outlay and treatment costs of the sands plant are much less than those of the slimes plant it is evident that a combination of the two plants, sands plant for sands, and slimes plant for slimes, is the best for our purpose.

In all the work done here I find that when operating under regular conditions the percentage extraction on any particular sized product is a fixed figure which is quite independent of the charge value. From this it follows that, given the percentages of the various grades, one can calculate the percentage extraction for the whole sample, or, given the total percentage extraction or efficiency number one can, with the help of a factor, obtain a figure which should correspond closely to the efficiency number for the sample or the percentage extraction.

$$\frac{\text{Per cent. extraction}}{\text{factor}} = \text{efficiency number.}$$

$$\text{Efficiency} \times \text{factor} = \text{per cent. extraction.}$$

The factor should be obtained from a long series of grading results and assays, otherwise a material error may be introduced. These factors assist in locating errors and any serious departure from them should be at once looked into. At Redjang Lebong, work at present is in a transitory stage, the stamping and tube milling conditions continually changing as we increase our plant, so I have not been able to use the factors over any long period. I find that a fairly regular factor for the sand plant extractions can be obtained direct from the daily tube mill grading efficiency number.

For correct efficiency numbers it is necessary to use a large number of screens otherwise considerable errors may be introduced, but for general plant work a few standard sizes answer very well in comparing the work done and the distribution of values. Unless the percentage of coarse grades is high no great error is introduced into the calculation, but when we come to the fine screens small differences of the grades bring about large differences in the per cent. efficiency; thus the separation of the −200 product into sands −200, $\left(\frac{0{\cdot}003'' + 0{\cdot}002''}{2} = 0{\cdot}0025'' \text{ mean}\right)$ and slimes elutriated, $\left(\frac{0{\cdot}002'' + 0{\cdot}001''}{2} = 0{\cdot}0015'' \text{ mean}\right)$ in place of calculating on a basis of −200 only, $\left(\frac{0{\cdot}003'' + 0{\cdot}001''}{2} = 0{\cdot}002'' \text{ mean}\right)$ completely alters results. For instance, the June sands gradings given above, work out the one way 349·0 and the other 361·2, whilst the slimes gradings give 596·3 and 484·4, respectively. The stamp mill at Redjang Lebong consists of 70—1,150 lb. stamps. The stamp duty is as follows:—

16 mesh	20 mesh	35 mesh	
280	259	210	tons
183	183	183	h.p.
4·0	3·7	3·0	tons per stamp per day

There are six Schmidt-Davidsen tube mills, 15 ft. 6 in. × 4 ft. 2 in. inside the iron shell; only four of these are used. The mills are lined with a local eruptive rock, and are fed with reef pebbles. Each mill consumes on the average 47·5 h.p., and on starting requires considerably more.

As the result of a large number of gradings I obtained the following results:—

BATTERY SCREEN SAMPLES.

	+100.	+150.	+200.	−200.	Efficiency Number.
16 mesh ...	45·7	8·2	5·2	41·1	259·
20 mesh ..	43·2	7·9	6·2	42·7	271·
35 mesh ...	24·8	9·3	9·2	52·7	326·

AFTER TUBE MILLS.

Stamps	Mesh	Tube Mills	+100.	+150.	+200.	−200.	Efficiency Number.
70	16	3	4·8	23·8	8·5	62·9	393·
70	16	4	1·1	7·2	17·7	74·0	438·
70	20	3	3·4	13·8	14·4	68·3	415·
70	20	4	1·3	7·1	14·0	77·5	444·

Month of June, 1908. SANDS GRADING SAMPLES. Nine days cyanide treatment.

	+100.			+150.			+200.			SANDS −200.			ELUTRIATED SLIMES.			GENERAL SAMPLES.		Efficiency Numbers.
	Per Cent.	Gold. dwt.	Silver. dwt.	Per Cent.	Gold. dwt.	Silver. dwt.	Per Cent.	Gold. dwt.	Silver. dwt.	Per Cent.	Gold. dwt.	Silver. dwt.	Per Cent.	Gold. dwt.	Silver. dwt.	Gold. dwt.	Silver. dwt.	
Charges ...	6·1	14·8	100·7	29·5	15·6	111·4	13·6	15·0	110·0	36·0	21·4	95·1	14·7	20·4	175·6	18·2	114·0	356·3
Residues ...	5·6	5·1	51·9	28·9	4·6	37·4	13·5	2·8	30·2	35·9	1·8	25·7	15·9	1·6	17·9	2·89	2·89	363·7
% Extractions	5·8	65·5	48·4	29·2	70·5	66·4	13·6	81·3	72·5	35·9	91·5	72·9	15·3	92·1	89·8	84·1	93·8	359·7
															Factors :	0·234	0·205	

June 1-15 and 16-20. Mixing 17·1 hours. SLIMES GRADING SAMPLES. Presses = ½ hour filling and ½ hour washing.

	SANDS +200.			SANDS −200.			ELUTRIATED SLIMES.			BEFORE WASH.		Efficiency Numbers.
	Per Cent.	Gold. dwt.	Silver. dwt.	Per Cent.	Gold. dwt.	Silver. dwt.	Per Cent	Gold. dwt.	Silver. dwt.	Gold. dwt.	Silver. dwt.	
Charges ...	7·3	12·3	91·4	15·9	15·5	128·5	76·7	16·9	159·8	16·38	149·6	484·4
Residues ...	8·1	3·2	43·8	18·6	1·9	37·6	73·2	·5	17·0	1·35	28·7	482·8
% Extractions ...		73·9	63·0		87·7	70·7		97·0	89·3	91·7	80·8	483·6
									Factors :	0·189	0·167	

In placing our tube mill efficiency numbers side by side with published results from other mines, it was suggested to me by Mr. S. J. Truscott that the introduction of the power figure and the tonnage would make the comparison fairer, as power and output apparently varied so much in different localities. This gave a figure which we called the relative efficiency :—

$$\text{Relative efficiency} = \frac{\text{Grading efficiency No.} \times \text{tons.}}{\text{Horse power.}}$$

It will be seen that it affords a fair means of comparing all types of crushing and grinding appliances, and should, I think, prove useful in assisting in the selection of the most efficient and economical machine.

Let us glance at the crushing figures and see how they compare :—

Stamps.	Screens.	Tons per stamp per Day.	Tube Mills.	Tons per Day.	Horse Power	Grinding Efficiency.	Relative Efficiency.	Relative Efficiency to Unity.
70	35	3·0	—	210	183	326	373	1·00
70	20	3·7	—	259	183	271	383	1·02
70	16	4·0	—	280	183	259	396	1·06
70	20	3·7	3	259	325	415	330	·88
70	20	3·7	4	259	373	444	308	·82
70	16	4·0	3	280	325	393	341	·91
70	16	4·0	4	280	373	438	328	·87

The table discloses that stamping with 35 mesh screen compares favourably with the coarser screens, and that when running tube mills in conjunction with the stamps the relative efficiency is not so high as with stamps alone, but there is a tendency for an increased efficiency with the former when coarser screening is used.

I regret that I have not been able to find any complete grading results of work done in pans, Chilian mills, Huntington mills, etc., so I cannot compare the relative efficiencies of these machines with stamps and tube mills, but perhaps some of the members will oblige with the necessary figures, as a comparison of these machines would be most interesting.

The following extraction results were obtained with the above mentioned crushing conditions (see table next page) :—

The falling off in the slimes extractions here exhibited is due to the increased tonnage handled reducing the available mixer time, to the presence of fine sand which really requires an extension of the mixer time, and to the presence of fine sand in the filterpress cakes rendering the washing more imperfect.

I trust that these few notes, written to re-open discussion on a most interesting phase of metallurgical work, will attain their object, and

Crushing Conditions.			Amalgamation Per Cent.		Sands Extraction.		Slimes Extraction		Slimes. Mixing. Time.	Proportions of Sands and Slimes.		Sands and Slimes Extraction.		Amalgamation and Sands and Slimes Extraction.		Efficiency Numbers.
Stamps	Screen	Tube Mills.	Gold %	Silver %	Gold %	Silver %	Gold %	Silver %	Hours	Sands %	Slime. %	Gold. %	Silver. %	Gold. %	Silver. %	
70	35	—	10·0	1·0	77·76	72·82	94·6	88·5	20·2	60·5	39·5	84·5	79·1	86·0	79·3	326
70	16	3	12·0	1·2	85·41	77·18	91·4	80·6	16·5	52·8	47·2	88·2	78·8	89·6	79·0	393
70	20	3	12·0	1·2	85·6	77·7	92·8	81.9	17·2	49·5	50·5	89·2	79·8	90·5	80·0	415
70	16	4	13·2	1·5	86·7	78·7	91·2	76·5	15·6	43·5	56·5	89·2	77·4	90·6	77·8	438
70	20	4	13·2	1·5	88·5	81·4	92·9	80·7	16·7	42·2	57·8	89·4	79·4	90·8	78·8	444

that members will favour the Society and myself by expressing their views on the points I raise.

The President: We must pass a very hearty vote of thanks to Mr. Yates for his very interesting paper on this subject. It is a very much debated subject as to which is the best fine grinding machine, and I hope that any of our members who have any data, will give us the results of work done with the Chilian and Huntington mills or with any other fine grinding machines, so that we may have an opportunity of comparing them with the author's results. It certainly is a very difficult thing to arrive at a correct estimation of the work actually done by tube mills, and I am sure that all of us who have anything to do with tube mills will welcome any data on the point. We are always trying to arrive at some method by which we can obtain the efficiency of tube mills, but, no doubt, many others like myself realise that when we get to a certain point we are metaphorically brought up against a brick wall and find our theories fall to the ground. We cannot go on and compute with any certainty, and I am sure that anything that will elucidate this point will be of very great benefit to the metallurgical work of these fields. It is a paper which calls for a good deal of thought.

I will now ask Mr. A. Thomas to give his reply to the discussion on his paper.

BATTERY AND CYANIDE GOLD SMELTING.

(Read at July Meeting, 1908.)

By A. Thomas (Member).

REPLY TO DISCUSSION.

Mr. A. Thomas: In reply to the criticism of Mr. H. A. White, I certainly recommend water cooled bearers for retorts on account of their durability; to my knowledge, the bearers at the Simmer and Jack Proprietary mines have been in use for 11 years, and are likely to last for the life of the mine.

It may be interesting to mention that 1,176,655 oz. = 40·3 tons of amalgam were treated in two retorts recently replaced.

Mr. White remarks that my description of the treatment of gold slimes is illogical, and again raises the question of the "pot" versus "Tavener" methods of smelting. It was not the intention of the writer to discuss the advantages or disadvantages of these methods, as I think there is quite sufficient subject matter there for a separate paper. As Mr. White has brought this matter up and has endeavoured to prove the Tavener method of smelting to be superior to any other known method, and makes his assertion chiefly on the hypothetical 3% loss of gold by using the pot method, I should like to state that my experience (and I am sure it is the experience of all who have experience of present methods of smelting gold slimes), is that the loss mentioned above is an imaginary one, and is most probably fathered by Mr. White by reason of his adherence to results obtained in the back ages, as witness his reference back to old numbers of our *Journal* to learn our present practice of smelting. It seems to me that we all have learned a little more about both methods of smelting since the 3% loss statement was made by one of our Past-Presidents some years ago, when the Tavener process was discussed by the members of this Society. The chief advantage claimed for the process was the cutting out of the acid treatment, but from what I can gather, the Tavener method has not fulfilled its function in that direction. I am quite prepared to admit to Mr. White that the flue losses are only a portion of the total losses under the pot method, but I consider it is the portion which is most liable to cause the greater loss. We have had figures before us which prove the loss at much below 3%. From my own experience of both methods of smelting, I consider that the "pot" method, as carried out at the present day, would compare very favourably with the "Tavener" method, provided only experienced men are entrusted with the work, when no difficulty would appear. I am quite at one with Mr. White in his contention that the assayer should have nothing to do with the smelting on account of the danger of contamination.

Mr. Bevington gives the dimensions of the retort flues he is using, and which are slightly less than those given in my paper. He says he has found from experience that it is advisable to reduce the area of the centre flue in order to check the drawing towards that point. This may be correct. I should like our President to give us his fuel consumption per 5,000 oz. amalgam retorted, and then we should probably learn which flues are the most suitable, given equal conditions, such as, the quality of fuel, area of grate, size of chimney, etc.

Mr. L. J. Wilmoth's discussion on the valuation of silver in by-products is very interesting. As he points out, the estimation of silver values in by-products, such as mentioned in my paper, requires special treatment in order to obtain true values.

With reference to my statement regarding the necessity of condensing chambers in the flues of furnaces for smelting rich gold products, I am sure this Society is indebted to Mr. A. McArthur Johnston and the Consolidated Goldfields Laboratory for the results of tests made with a view to discovering the actual loss of gold carried through the flues during a bi-monthly cyanide smelt. To my surprise this was valued at 7s. I had thought it to be much greater. I agree, however, with Mr. McA. Johnston that, judging from the proportion of gold and silver in the bullion recovered, it appears to be evident that the loss is caused by volatilisation. I have no doubt that the present method of fluxing gold slimes has to some extent eliminated mechanical losses of gold. It would seem that the flue losses are not so great as they have been in the past, and it is gratifying to know that we are making progress in this important branch of our industry.

With regard to the purple colour varying with the richness of the slimes, I am somewhat disappointed that we have heard nothing more definite from our analytical members, but trust that this subject will be brought up again at some future date, when we may obtain some enlightenment on the subject.

I note that Mr. A. H. Hartley's records of losses of mercury differ somewhat from mine, which may be accounted for by reason of the amalgam which he retorts containing a larger percentage of water.

With reference to Mr. Bevington's remarks, and to his quotation from Donald Clark's "Australian Mining and Metallurgy," I have never tried an oxidising agent while retorting amalgam, as recommended by Clark for impure amalgam. Another extract quoted by Mr. Bevington from the same work is:—"When impure gold is melted it is practically impossible to purify it by mere melting, even with the addition of fluxes." I give an instance from my own experience which has some bearing on the above statement. A bar of 587 oz. was brought up from 775 to 822 parts fine gold per 1,000, as follows, the principal impurity being copper:—

I placed the bar back into the pot and, when melted, stirred in a mixture of permanganate of potash, borax and sand. This operation was repeated three or four times in the course of an hour, bringing up the fineness of the bar, as stated above, to 822.

In reply to Mr. Bevington, I would state that I have had no experience with impure retorted gold, that is to say, below 800 fine; but as our President has informed us that a considerable amount of impure material has to be dealt with on small mines, and as the handling of such has not been touched upon by the author, I would granulate the base bullion, flux it with permanganate of potash, then resmelt and repeat if necessary. The necessity of keeping the unsmelted metal under the influence of the flux is the subject of another quotation from Clark by our President, who asks if this is correct procedure. Keeping the impure retorted metal under the influence of the flux prevents the escape of the available oxygen, and forces it to take up with the base impurities in the bullion, and, as the retorted bullion is porous, affords a large area for the oxidiser to operate upon. I should therefore assume that the procedure is not incorrect.

Prof. Stanley's remarks are most interesting, especially the sets of experiments which have been conducted by noted metallurgists, showing the losses of gold in alloys in different proportions and at varying temperatures. He also says that convincing experimental evidence is still wanting to justify the erection of dust catching chambers but further asserts that attention to this matter is essential.

Although Mr. McArthur Johnston's figures show this loss of gold to be small, I think it would be wise to err on the safe side and instal dust catching chambers, especially in such important work as gold smelting. The cost of these should not be very large.

Prof. Stanley's experiments in smelting gold slimes direct with bi-sulphate is interesting from a laboratory point of view, but I fail to see that the process could be worked advantageously. The Professor tells us that the best results were obtained by using two parts bi-sulphate, one half part borax, and one part of slime.

As an experiment, this process is interesting, but commercially it would be impossible. He further states that the process could be carried out on a large scale with clay liners and crucibles, or a small reverberatory furnace (by which I gather he means a pan furnace). In so doing, he would collect the gold with litharge, and cupel the lead bullion, which is practically the same as the Tavener process, plus the bi-sulphate. If the gold is not collected with lead in the pan furnace, I am curious to know how the Professor proposes to get the gold out from the furnace, as I think it would be a difficult matter to tap it out as is done with lead bullion.

In conclusion, I desire to thank those members who have taken part in the discussion, and trust that my few notes, and the discussion thereon, have been of some assistance in bringing our knowledge of smelting practice on these fields more up-to-date.

The President: On your behalf I thank the author for his excellent reply to the discussion on his paper.

I know that we have present to-night several members who are prepared with discussions on Mr. Weston's paper on the "Theory of Blasting with High Explosives," and as it is an extremely interesting subject I propose, with your consent, to proceed now with the discussion on that paper, and I will ask Mr. Leslie if he will be good enough to open the discussion.

THE THEORY OF BLASTING WITH HIGH EXPLOSIVES.

(*Read at October Meeting, 1908.*)

By E. M. Weston, A.S.M.B. (Member).

DISCUSSION.

Mr. C. D. Leslie (*Member*): The great necessity for rigid attention in the use of explosives has to be thoroughly recognised sooner or later, and I trust that the author's valuable and carefully compiled contribution will lead to a full understanding of the position as affecting our mines.

I do not now propose attempting to reconcile the "theory" of blasting with the "practice" on these fields, but wish to call attention to certain practical considerations, the neglect of which would appear to prevent us from realising, and consequently from determining, to what extent improvement is possible, and what far reaching results would accrue from improvements in the use of explosives.

I hold the opinion that the questions of health and efficiency in underground work (and everyone should recognise the beneficial effect of health on efficiency) are dependent on the explosives question, and that the great remedy for both should be sought for, first, through the expert use of explosives. Although it is difficult even to approximate the amount of explosives used on these fields in excess of the requirements, many of us are convinced that, besides entailing a large unnecessary expenditure, it adds materially to the noxious gases generated in our mines, and in a greatly increased proportion to the noxious gases liberated from a like quantity of explosives against which full burden has been exerted. The remedy for the whole situation is evidently to be sought for in properly confining in drill holes the explosives used in our mines, and also in having full burden exerted against the explosive. It would then be conceivable that the cost of breaking rock (which may be taken as the determining factor in working costs) would be reduced and the number of units employed in breaking rock would be decreased. This should mean :—

1. Fewer working faces.
2. Less development ahead.
3. Less supervision.
4. A reduced area to be ventilated.
5. An improvement in health conditions.

Prove to the miner that he can break rock from the drill holes he is in the habit of planning, with less explosives than he is in the habit of using, and on his part will follow the inclination to increase the burden, believing it can be broken; and, as he learns that the burden can be safely increased, the value of the longer drill hole will, I believe, also become apparent, for reasons to which I shall presently refer.

In stoping, the miner becomes very doubtful with regard to the amount of work a given quantity of explosives will accomplish, if, after blasting, he finds that the rock at the collar "comes," leaving a socket; but this may be the result of an unbalanced hole which, either from being too short, or because it "looks in," gives the easiest release towards the collar, and consequently breaks upwards. It must not, however, be forgotten that a bench frequently breaks upwards, leaving a socket, because the drill hole is overloaded with explosives. Errors with regard to the general direction of drill holes are perhaps due as often to carelessness as to want of skill (the miner having his off day like other people, which is often not to be wondered at when conditions are taken into consideration), but I believe that the value of the longer hole is not generally appreciated.

In general practice on these fields it would probably be profitable to have less regard for the danger of "bull ringing," and pay more attention in stoping to breaking out benches obtusely at the bottom where the burnt ground from the former blast helps the release. In this connection it would be well to study the "grain" of the rock to be broken and have regard to certain cleavage planes or "cleats"—often indistinct—which, so far as I have observed, trend from "foot" to "hanging" throughout our mines (certain disturbed areas excepted), and which occur more or less in parallels, usually intersected by joints. "Break rock with the grain" is an axiom amongst skilled miners.

In stoping, my opinion is that mine rock on these fields breaks so much better with the grain than against it, as to warrant drilling a longer average hole than is now the practice, and "with the grain." Longer stope benches so drilled minimise the number of times that rock, against the cleavage—where the greatest resistance is offered—is broken off; the more so, since the longer drill hole gives the incentive to increase the burden—if only to prevent "bull ringing" from too easy release from explosives towards the bottom of the hole. The fact that "bulling," as a consequence of blasting holes which appear to be well balanced, is commonly the result of drilling against the grain, will furnish a strong argument against making back-stoping a practice on the Witwatersrand, if proof is forthcoming that pronounced cleavage planes exist everywhere in our mines, intersecting the planes of stratification at approximately right angles to the strike and indicating the grain of the ground. This theory has been advanced by Mr. Musson Thomas, of the Jumpers Deep, who is now engaged in introducing a system (which is being watched with great interest) of drilling parallel to these planes. The idea is, that if the burden is suitable, the hole should be drilled in front of, and parallel with, the north and south cleavage plane, not only to prevent "fitchering" by the point of the drill entering the cleat, but also to allow explosives to break back to it.

The appearance of stope benches after blasting tells its tale. If they are ragged, and the broken ground lies close to them, it is evident that economy with explosives has been considered. It is pleasing to note that this occasionally happens, even though the miner is fully aware that by practising economy in the use of explosives, he gives himself or his partner a considerable amount of work on the following shift in "dressing down" the ragged stope benches. Economy so exercised with explosives, and the consequent reduction in the quantity of injurious gases, combined with the physical exertion of "dressing down" might be looked upon as a preventative of miners' phthisis. Men who inhale explosive fumes are "put off their feed" and become unable to pursue their work energetically, the consequence being that their lungs, not being fully exercised, become receptive to silicosis. In hammer work the loss in efficiency from listlessness on the part of the miner and his natives, due to bad air, is well exhibited in an up-cast as compared with a downcast stope, and the improvement with hammer boys in a downcast and right handed stope as compared with an up-cast and left handed stope, is sometimes very apparent.

There is further improvement if drilling natives are carefully classified in stopes according to the drilling inches that may be expected of them, and also if drills are gauged so that explosives can be pressed home to the bottom of the drill hole. It is very important that natives should be specialised to drill in one class of ground, carefully avoiding transfer to a different class, especially where one reef is known to be harder to drill than others in the mine. If increase in the length of drill holes is advisable it is essential to good results to preserve the length of stope benches, and it could not be considered good practice in hammer stopes to destroy benches weekly by undertaking the shorter holes which would be drilled during the shorter Saturday shift.

Reference might here be made to the excessive waste in explosives through cutting stope and drive pillars, giving tight ends everywhere, and deforming the stopes, as well as losing, for the time being at least, the reef contained in these pillars. The remedy for this evil may be sought for in the substitution of the "pack" for the customary pillar, especially where such pillar scales and shapes "like a lady's waist," under the enormous pressure per square inch it—the pillar not the lady's waist—is subjected to in deep level mining, and also in increasing the distance between levels, particularly where the dip of the reef is flat. If the short Saturday shift is worked, it would seem advisable to use that day as much as possible, so far as hammer boys are concerned, in shaping benches which have become shortened or deformed. In hammer work, if the native is unable to drill a long hole well within the daily shift, he will frequently fail to get the required length. This, and the cutting of pillars are customary reasons for losing the succession of proper benches, and there is considerable attendant loss in continually re-shaping them. The miner, uncertain as to the length of hole the native may manage to drill, will effect a compromise with regard to burden, and the resultant loss in efficiency can be imagined. It is here that the small stope drill, when it is perfected, will count, the miner making a more accurate estimate of burden by being comparatively sure of the direction and length of hole. If it is found impracticable with a small stope drill to restrict the diameter of the hole to a size which $\frac{7}{8}$ in. explosives will fit, I suggest the manufacture of 1 in. explosives rather than the use of the $1\frac{1}{4}$ in. sometimes employed in blasting these holes.

If, in longer drill holes, explosives can be better confined, the steel question has to be considered. The profits derived from the use of improved drill steel, with improved methods of sharpening and tempering and due precautions as to length and gauge of succeeding bits, are probably so great as to entirely outweigh the consideration of the increase in cost of steel and its preparation for daily use. It seems ridiculous to undertake the huge expenditure required to develop and equip our mines and to organise a labour scheme, etc., and finally fail at the point of the drill, where the difference in results can be made so important. If, through the use of improved steel, higher air pressure, etc., the quantity of rock broken daily at each working face can be increased, it would be found that, either there are enough working faces in the mine to permit of single shifting, or there need be less anxiety than exists at present to advance development work, which could probably be done at a cheaper rate when the ground will be required for stoping in years to come, very much cheaper when interest, which would be received on the deferred disbursement on development, is added to the capital available for this purpose.

I am glad that the author has drawn attention to the limit depth of hole laid down for the coming stope drill trials. I believe that a maximum length of 48 in. does not adequately provide for the relation which depth of hole bears to stoping width.

Mine conditions, as regards air pressure, steel, etc., do not seem good enough for such an important contest, and I consider that each competitor should be encouraged to make his conditions as favourable as possible, if only to demonstrate to what extent small stope drills can be made profitable. If one of these competing drills is able to deliver a blow which the steel of the mine will not "stand up" to, the machine will be put out of the running for the prize. Surely the thing to do, if it were not the machine that failed, would be to get steel which would stand up to the work required of it.

The author's suggestion to substitute a 5 lb. hammer for a 4 lb. hammer for drilling boys will not meet with general approval. A number of our mines now use a 4 lb. hammer head which, instead of being short and having a broad face, as in the old style of hammer used on these fields, is elongated and faced down to the size of the drill heads. The more direct blow, as struck with this hammer, is more than equal to adding 1 lb. to the weight of the old squat hammer head.

In charging up holes, lack of time taken to do justice to the work, has much to do with overestimating the quantity of explosives required. Low air pressures, inferior steel, and sometimes lack of the energy and system required in starting up machines or hammer boys with the least possible delay, retard the completion of drilling work until the miner has to make up time by rushing the work of charging up. It should never take less than $1\frac{1}{2}$ hours to efficiently charge and stem 30 hammer holes, yet this work is generally done in less than an hour. Needless to say, the work is not properly done, and it is small wonder if, in consequence, the miner perforce distrusts what can be accomplished with explosives.

In stoping, with large machines, the drill holes are, I believe, generally too short for the diameter of the explosive used, and the uncertainty with regard to what back holes will do, is responsible to a large extent for the increased quantity of explosives used per ton broken by machine, as against hand labour. The uncertainty as to how far front holes will "break back" has fostered the bad habit of staggering explosives in back holes by interposing tamping, with the idea of preventing "bull ringing" by distributing the shock, in the event of the burden towards the collar proving greater than towards the bottom of the hole. Until the principle can be established of drilling holes which are not dependent on each other, this disadvantage would be largely overcome by drilling only front hole benches with machines.

The additional rigging up and down, which such work would entail is an argument against the use in stoping of the present heavy $3\frac{1}{4}$ in. rock drill.

Too much paper around explosives, and paper that is too stiff, are disadvantages. The tucked in ends of such paper hold when the tamping rod is pressed against the cartridge, and prevent the cartridge bulging to fit the drill hole. That may be a reason why certain miners see no harm in "chucking in" cartridges two at a time, and adopting the same process with regard to tamping cartridges. Something might be said regarding incomplete explosion throughout the charge, because of the intervention of the thick wads of paper—two at each joint—formed by the tucked in ends of the cartridge wrappers.

The efficient charging of drill holes is important, and much can be gained by the use of a well turned copper headed tamping rod, when compared with the work done by the usual light, roughly made rod, which becomes rounded at end and forms a concavity in each cartridge being "pressed home." The cost of preparing tamping on the surface is less than the cost of employing natives underground for this purpose, and a much better tamping can be prepared on the surface than that made underground with mine dirt. Fine slimes, as suggested by Mr. Nitch, of the Meyer and Charlton mine, make an excellent tamping, and tamping paper can be cut and rolled in the same manner as the paper used for explosive cartridges, instead of being cut square and held together with candle grease, as is so commonly done underground. With proper appliances, a native can fill 1,000 cartridges per day, while underground he would fritter away the best part of half a shift in preparing 50 to 60 tamping plugs which are bad ones at that. Experience has shown that improved tamping holds and is not displaced in drill holes by blasting to the same extent as is found in the case of tamping made with mine dirt.

So much ground has to be covered in dealing with the important subject introduced by the author, that these somewhat disjointed remarks are made, more with a view to adding to the considerations involved, than of confining attention to a criticism of his excellent contribution.

The President: I think we are indebted to Mr. Leslie for his very valuable contribution to the discussion. It is a matter which I know he has taken a very great interest in, and given a great deal of thought to, as you will gather from the remarks he has made.

Mr. H. Musson Thomas (*Member*): I have read with much appreciation the able paper which the author has contributed.

I do not, at this stage, intend to offer any criticism, but would like to contribute a few notes which might be termed introductory to the author's paper. These notes are part of a small treatise which I have prepared for the guidance of my underground staff.

Grain of the Witwatersrand Strata.—Until the foundation of efficient breaking is ascertained, I do not think much progress will be noticeable in breaking ground by first determining the burden, length and diameter of hole, and quantity of dynamite for a particular stope width, The real crux of efficient breaking is to discover the tendency of the conglomerates and the country rock to break more easily in one direction than in any other; in other words, to find the 'grain' of the rock, expressed in miners' phraseology. Although the illustrations show benches for machines, the principles apply with equal force to hammer benches. They also apply more particularly to stopes in which the dip is less than 40°. A very common question is:—Should a particular stope face be laid out for overhand, underhand, or breast stoping, or a combination? I hope to be able to answer this query.

It is generally accepted that the direction of the face should facilitate shovelling, and this latter point is allowed to bulk so much in the mind of the overseer, that it not infrequently happens that the direction is determined by the facility for shovelling only.

Before proceeding further, it gives me much pleasure to state my full appreciation of the good work that has been carried out by a large proportion of the skilled miners working on these fields. I allude to those miners who plan their benches in relation to the grain of the ground. To these miners I would suggest, if I may be allowed, that having found the grain that I allude to, in one stope, the same grain will be found in a similar direction in any other stope in either the same mine or any mine on these fields unless planes caused by local disturbances are better defined.

The full significance locally of the occurrence of fracture planes is not generally appreciated and has not, to my knowledge, been thoroughly systematised in any of our mines. I would, therefore, be much obliged to those miners who may have worked on similar planes in other parts of the world if they would give us the benefit of of their experience.

The tightness of rock or the resistance to breaking is due to the following reasons in their order of importance:—

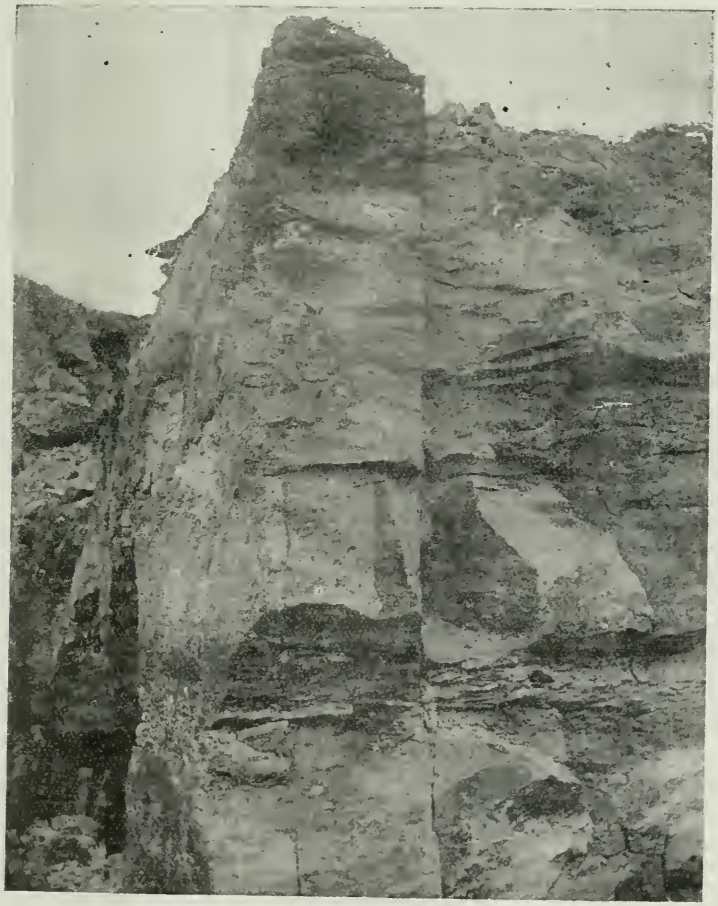
(1) Drilling across fracture planes or lines of weakness.

(2) Rolls or curvings in the hanging and footwall.

(3) A tight or frosted hanging and footwall as opposed to a loose "let go."

(4) The toughness of some rock as opposed to the brittleness of other rock.

Drilling across Fracture Planes or Lines of Weakness.—It will be found on examination in mines on the Rand that there is a series of vertical parallel planes which cut right through the reef and waste, but do not throw the reef, and run more or less north and south, at right angles to the strike of the strata. For terms of reference, these planes might be called "fracture planes." My reason for assuming that these planes will be found in mines of the Rand is that I cannot trace any local origin or cause for such planes in the Jumpers Deep. After having found well defined cracks in a mine the first idea would be to trace the origin to the fault or dyke which has caused the largest throw and see if these cracks were not running either parallel or at right angles to the strike of faulting. The largest throw has been caused by the Heriot dyke, which runs in a north-easterly direction through the mine, but the strike of the cracks is more or less north and south. In the same way, other throws might be cited all of which are not sympathetic to the fracture planes. It has been suggested that the planes were induced during the general upheaval of the Witwatersrand strata, and that the planes are younger than the faults and dykes. Such a suggestion is very plausible; but I do not intend to trespass into the regions of geology. I trust that some of our geological colleagues will explain the origin of these interesting planes. Through the courtesy of Mr. C. D. Leslie, I have photographed fracture planes and lines of weakness, which are very pronounced in one of the outcrop workings of the Simmer and Jack mine. A copy of this photograph is reproduced.



Photograph showing "Fracture Planes" on the Simmer and Jack Mine.

The conglomerates have not been tilted up uniformly throughout the Rand. The tilting varies from a vertical dip to a very low dip. Consequently the conglomerates were subjected to unequal upheaval forces. For the sake of an example take half a mile along the strike of the present outcrop. It is reasonable to assume that, if at the eastern end of the half mile the tilting became vertical, and at the western end it was 20°, that the unequal forces would be so pronounced as to produce cracks in the strata at right angles to the tilting strike. Again, if a certain pressure were sufficient to produce cracks, it is also natural to assume that there was a tendency for cracks or fractures to be formed in between two fracture planes. This tendency would produce embryo fracture planes which are invisible to the eye and which might be termed lines of weakness. These lines of weakness are well pronounced after blasting, and they naturally run parallel to the fracture planes. I have also noted two other series of planes which strike at approximately 45° to the fracture plane on either side of this plane, but, fortunately, they are not so well defined nor so universal as the fracture planes. Faulting planes which throw the reef, but do not necessarily run parallel with each other, must not be confused with fracture planes.

I shall now endeavour to suggest the way in which fracture planes may be used to produce increased efficiency in all branches where rock is broken. Without taking fracture planes into consideration their existence will undoubtedly hinder efficiency.

Stoping.—Of the six sides of a bench, four are bound and the other two are free. The section at right angles to the direction of the holes which produces the greatest line of resistance is at the base of the bench, where the whole of one side and the part of two sides have to be sheared, and part of one side has to be rended. Consequently the greatest force is required at the bottom of a hole. If the bench could be so arranged that the fracture plane coincided with the base of the bench, this plane would thus be made to help blasting to the greatest extent. Figure AA shows bench (1) arranged in the ideal way. Unfortunately there are two non-existent conditions which are essential for the successful carrying out of this principle. The first essential is that the rock should have a massive structure. It should be like putty, with no cleavages at all but just one homogeneous mass. On the other hand, our rock is, apart from bastard slips, intersected with invisible lines of weakness, which, although not so definitely cracked as fracture planes, are still cracked to a smaller degree to such an extent that if microscopically examined would show a distinct cleavage. The resultant action, when under the influence of an explosive charge, would be similar to that of the fracture plane. I have seen these lines of weakness very pronounced in as small a distance as every 3 in. If one of these lines were located as shown in Figure AA, bench (1), cutting through just above the bottom cartridge, the base of the hole would undoubtedly be held up.

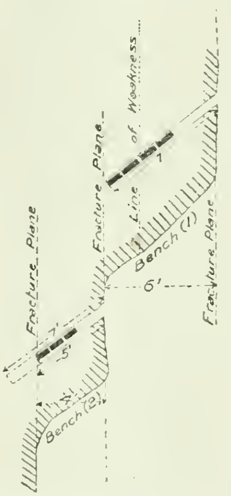


Fig. AA.—This principle would break the greatest fathomage if local conditions were favourable.

The second essential is that, besides the rock being a homogeneous mass, the lines of cleavage should be spaced at equal distances, say, 6 ft. apart. This condition is also non-existent, with the result that although the hole could be put in the correct distance of, say, 7 ft. in bench (1) there would be a greater chance of the fracture plane on the next bench appearing as in bench (2), when either the last 2 ft. of the hole would be lost or this hole would only be able to be put in to a distance of 5 ft., which would at once spoil the system of the benches. I consequently consider that these planes should be made use of as in Figure B, which would appear to produce the greatest efficiency.

Before explaining Figure B, I shall demonstrate how inefficient breaking will result, if these planes are not taken into consideration.

The holes in Figure A run in one case diagonally across the fracture plane and in the other case at right angles. In Figure A (1) practical knowledge will agree that holes (a) and (b) will be held up at their bases. If I may venture the reason it would be that as the fracture plane intersects the second dynamite cartridge in hole (b), this breaks up the total effect of the explosion into the largest force above the plane and the smallest below the plane. The dynamite above the plane has to perform two duties. It has to push out the rock above the plane, and also to help the explosive force of the dynamite below the plane by exerting a pull on that portion of the rock. There is, therefore, no beneficial pull produced by the excessive dynamite above the plane on the rock below the plane, since a cleavage destroys the action, and the one and a half cartridges have not sufficient force to push out the burden on the bottom of the hole by their own agency. Similar remarks apply to hole (a). The resultant bench would be represented by the

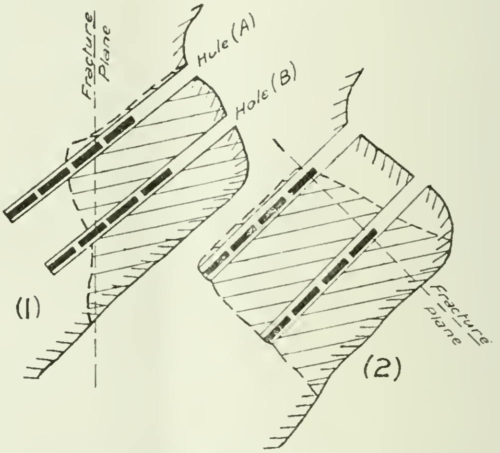


Fig. A.—Holes drilled across Fracture Planes or Lines of Weakness are either held up at the collar or base and result in inefficient breaking.

dashed line. In Figure A (2) there is a line of fracture near the collar, which produces an easy "let go," and there would most probably be a line of weakness near the bottom of the holes, so that the resultant bench would be as shown by the dashed line, and the collar would be left intact, causing a bull-ring. In Figure B, (hole c) is put in parallel to a fracture plane, which may be seen, and there would most likely be a line of weakness running parallel with the fracture plane in or near hole (d). In this case the direction of the hole and fracture plane being similar, it follows that there is an equal cohesion of the rock all along the length of the hole, which produces a constant burden, which in turn assures that an equal quantity of explosives will break a similar burden on all such placed benches. I would invite the opinion of members as to whether a hole should be put in along the fracture or on one side of the plane and parallel to it. In the first case the explosive would not have to rend the rock along the plane, since the rend has been produced by the fracture, and therefore it might be assumed that a certain amount of dynamite would be saved. It has been proved that gelatine creates a pressure on explosion of about 20,000 lb. per sq. in., and it is not hard to contemplate that with this huge pressure there would be a leakage of force along the crack although the sludge in a wet hole would act as tamping. Another objection is that the fracture would cause the chisel bit to fitchure. I see no reason why three cartridges should not only break as much as the four in Figure A, but more, since the broken bench would be represented by the dashed line, and the bench would approximate the much desired square instead of the narrow rectangle.

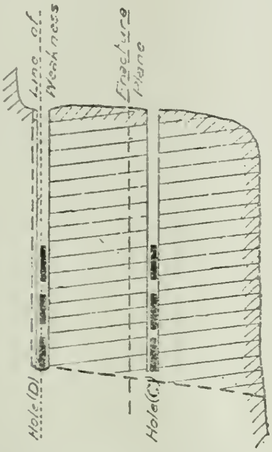


Fig. B.—Holes drilled parallel to Fracture Planes and Lines of Weakness will break the greatest fathomage under local conditions.

Hole (d) shows no fracture plane before it is broken, but if the hole is put in parallel to the direction of the visible fracture plane it will be found, as a rule, that there is a line of weakness which, after breaking, looks similar to the fracture plane. It follows, when a fracture plane cannot be found, look for a line of weakness on the broken bench and they will be found to be more numerous than fracture planes. I do not mean to infer that, even if the above suggestions are minutely carried out, every bench so drilled will break efficiently. Besides these planes there are bastard slips running in no particular direction but which nevertheless can be seen on benches which show well defined fracture planes. These bastard slips must also be taken into consideration, and if more pronounced than the fracture planes would naturally be followed in the place of the above-mentioned planes.

I think the reason why it has been found necesssary to give a lift on the bottom of holes—that is, to have less burden on the bottom of a hole than at the collar, will be explained by Figure A (1), and that if the hole is directed as in Figure B a lift will be unnecessary, and an equal burden can be placed on the bottom and the collar. I, also, do not think it will be found necessary to drill the two back holes, say, 6 in. deeper than the two front holes. It would appear that the reason of these extra 6 in. being drilled is caused by drilling across the fracture planes, as shown in Figure A (1).

In passing, I might draw your attention to the fact that stoping, according to Figure B, will produce fewer misfires and fewer accidents, as not so many bottoms of holes will be held up.

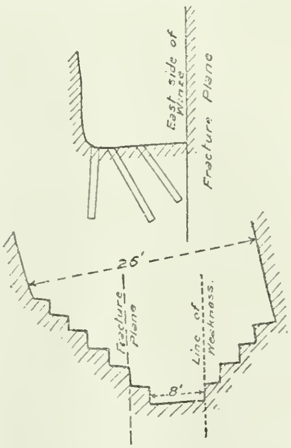
The answer to the question as to whether a stope should be run in an overhand, underhand or breast direction is:—Make the bench faces parallel to the fracture planes, or in a few cases to the bastard slips, and the stope faces will shape themselves accordingly. The obvious retort is:—"If the stope is run in an underhand instead of an overhand direction the shovelling costs will be increased." Let us assume that this system produces an increase in the shovelling costs of 25%, and a decrease in the stoping costs of a similar amount. If 1s. 6d. per ton is taken as a fair charge for shovelling and tramming the proportionate shovelling cost will be about 1s. Four shillings and sixpence might be cited as a fair cost for stoping. There would thus be a nett gain of 10½d. per ton, which on a 200 stamp basis, mining 40,000 tons per month, represents £1,750.

The desired end to be attained is the maximum duty in the minimum stope width. Experience teaches us that when a miner drills in the same direction as the dip of the stope there is less likelihood of the bottom of the holes being projected into the hanging or foot. This fact is generally proved when undercutting a pillar when, as a rule, the foot is ploughed up. Since the fracture planes and their sympathetic friends, the lines of weakness, run north and south the direction of drill holes will be south, or in the same direction as the dip, which will produce the minimum stope width.

Development.—It is interesting to enquire how fracture planes will help development.

The average drive is extended in a direction which is at right angles to these planes, so I am afraid that no benefit can be derived from them in drives running east and west. In raising, winzing, and incline sinking, they are, however, of distinct advantage. Since they run north and south, a winze should be started on a well defined fracture plane, provided this plane can be found at approximately the desired point. The winze would be arranged so that the plane coincided with the east face in order to not cramp the hammer boy, which would be the case if the west face and the plane coincided. This plane, if there were a well defined foot, would fulfil the functions of two sides of a cut; and the starting hole would be drilled at an angle to the planes so that its base would just touch, or nearly reach the planes. Similar remarks apply to a rise.

Since most of the inclines are sunk at an angle to the dip it is more difficult to use them, but it will be conceded that if, say, an 8 ft. winze were always kept ahead of the main face of a 26 ft. incline by small machines, and the other 18 ft. were benched out parallel to these planes, an increased footage would be obtained.



Winze and Incline. Hammer work.

Rolls or Curvings in the Hanging and Foot Walls.—In Figure C the dashed line ZZ represents the normal hanging, but the wall has been broken up by unconformable rolls. Figure D represents a plan of these rolls showing their strikes.

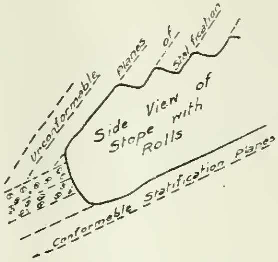

Unconformable planes of Stratification
Side View of Stope with Rolls
Conformable Stratification Planes

Fig. C.

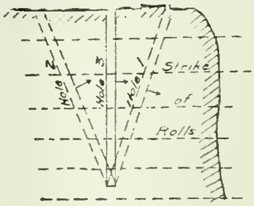


Fig. D.—Hole No. 3 would be most suitable for Rolls and Fracture Planes.

If hole No. 1 were directed as shown, there would be a tight end and it would not break efficiently. Holes 2 and 3 would not form wedges at their bottoms and would break more easily. Hole 2 would perhaps give the greatest efficiency. The fracture planes on the Rand generally run at right angles to the rolls, the latter of which strike east and west. Therefore hole 3 would suit both the fracture planes and the rolls.

In the case of a stope with a "tight" or "frosted" hanging, as opposed to a loose "let go," and also when the rock is *tough*, as opposed to *britlte*, for breaking purposes; I do not know of any method of producing greater efficiency by altering the system of breaking ground. Knotted ground also hampers breaking; it appears to have a spherical structure. I might illustrate this by directing your attention to those lumps of ground which prove so difficult to break by the native wielding a 12 lb. hammer at the grizzley of a main ore box. These are three natural conditions of tightness which we have to put up with, but the other two conditions can be made to help the breaking of ground.

Summary.—There are machine drills on the Rand which may justly be termed fool-proof in their mechanism, and with increased air pressures, single shift working, clay tamping cartridges, and by sharpening drills underground, there is only one other factor to be made fool-proof as far as possible, and that is, the actual breaking of the rock. It is natural that, if a miner who is learning to break rock were instructed to put his holes in one direction only, and that direction was parallel to the fracture planes and the lines of weakness, he would soon be able to judge the least amount of explosives for the largest burden. But if he is called upon to cut a pillar he will have to put in the holes across the planes which are running north and south, and he will be surprised that his holes have not broken efficiently in bench A Figure E. His confidence in himself will begin to waver, and by the time the pillar is undercut, all confidence in his ideas on burden will have departed. Then again, the stoping machines are supposed to be used for stoping only, therefore they should not be called upon to do what might be called development work in cutting pillars and boxholes, where such are necessary instead of waste packs.

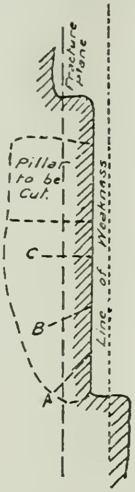
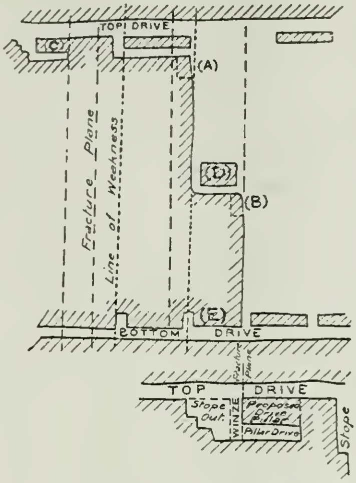


Fig. E.—It is wrong to undercut a pillar by back stoping, as the direction of the bench holes cut across the Fracture Planes and Lines of Weakness, resulting in inefficient breaking.

It is now essential to find some method of obtaining a uniform direction of hole, and also to keep the machines continually stoping. Figure F represents the method which I would recommend wherever possible.

The fracture planes and lines of weakness are striking north and south. In order to keep the machines continually stoping, and to enable the machines to always put their holes in parallel to the fracture planes, it is necessary to cut pillars and boxholes by some outside agency. The top drive pillar should be sunk through and cut under by hammer boys. The stope pillar should be undercut by hammers and sunk through by a machine, as hammer boys would take too long to sink through the pillar winze, thereby hanging up too much broken rock. The bottom drive pillar would be raised through by an independent machine or by hammer boys. The top pillar winze and the bottom pillar rises would be put up in advance of at least one pillar ahead. As soon as the hammer boys had undercut the top drive pillar and the stope pillar, the stoping machine would break into it and start drilling benches (a) and (b), and no holes would thus be put in at an angle to the fracture planes. I would suggest that where a bench face is not parallel to the grain, burden should be sacrificed on the front holes, and they should at once be directed parallel to the grain and not wait to procure the grain direction by gradually beating back, say, half a dozen benches. Consequently, if the learner is only asked to do elementary breaking of rock in one direction, I maintain that breaking rock is also reduced to such a simple process as to merit the distinction of being called "fool-proof," as well.



Sink winze and drive under proposed pillar (c), by hammers.

Sink through pillar (d) by machine and cut under by hammers

Raise through pillar (e) by machine.

The stoping machines will then always be "stoping" and the holes can be directed parallel to the Fracture Planes to produce the maximum fathomage.

TOP DRIVE PILLAR.

When framing this pillar, sink the winze on a Fracture Plane if possible. A certain amount of stoping could then be carried on from the winze which would benefit shovelling costs.

Fig. F.

It might be said that there is an increased cost in undercutting the two pillars by hammers, but if the increased efficiency of the machines be taken into account, I feel satisfied that when the system is working satisfactorily, a fair proportion of the stoping machines might be stopped, to produce the same amount of ground, and if such be the case, the gain of shutting down such machines will far outweigh the extra cost of undercutting pillars—if there be any. This system has now been in operation for some two months back, and in every case where the miner has been able to follow the principle, a marked advance has been registered, in breaking fathomage. I can consequently claim that these contentions have been removed from the realms of theory and have been fully demonstrated in actual practice.

In conclusion, I would refer back to a statement in the earlier pages that, until the grain of the rock, which will assist or retard breaking in a particular direction, has been ascertained, it is useless to make experiments to prove what the length of hole, diameter of hole, quantity of explosive, burden, etc., should be.

The President : Our thanks are due to Mr. Thomas for the very valuable contribution to this paper of Mr. Weston's. The utilisation of the theory of the fracture plane is, I think, somewhat unusual on these fields, but I know that Mr. Thomas has taken a very great interest in the matter, and has given a great deal of thought to it. We are glad to have his experience on the subject. Mr. Stuart Martin, whom I must welcome as one of our newest members, has also a contribution on this subject. I see from your applause that you agree with me in welcoming a gentleman of such eminence. It is very gratifying, I am sure, that we can obtain such men of experience amongst our members. We are here to learn, and desire to obtain all the information we can, on all the subjects taken up by our Society.

Mr. Stuart Martin (*Member*) : I should like to thank you for the kind way in which you have received me and made me a member of your Society. I can assure you that it is a great pleasure to me to become a member, and if there is anything that I can do to assist its progress I will gladly do it. As for contributing to this paper, I cannot say to-night that I am prepared with any contribution on the subject raised by Mr. Thomas. It is one in which I am keenly interested, and in which I should like to support Mr. Thomas, as, in my opinion, it is a very important point indeed. I have only been in this country some three months, and it is hardly yet time for me to come to any definite convictions or in any way to attempt to try and teach miners here how to break rock, as I know amongst your practical miners you have a large number of highly skilled men, and examples of men second to none in the world. The whole of my life I have been working in the collieries of the South Wales coalfield, and the most important and the first lesson a miner or collier is taught, in order that the coal may be worked skilfully is, the presence of lines of weakness, and how to take advantage of them. Now, in working the banket here, and the coal seams, there is a very great similarity. Your banket is very much harder, and blasting becomes an absolute necessity ; still, on the other hand, many of the coal seams cannot be worked to a profit without blasting, when disregarding the lines of weakness ; and in any case the efficiency is low when such lines of weakness are neglected. Referring to the central portion of the South Wales coalfield, where the best Welsh steam coals are worked, the lines of weakness are clearly defined, and are known as slips or cleavages, and are very regular. Many of you are aware that the conditions in the South Wales collieries are such that blasting is one of the last means a manager can resort to in breakground or working coal, owing to the presence of inflammable coal dust and fire-damp, and, therefore, blasting is not permitted except under special conditions. The result is that we have to work the coal skilfully, and take every advantage that nature provides, such as fracture planes and lines of weakness ; this also applies to the ripping of roof and sides in the country rock. It may be argued that when lines of weakness are observed in the stoping in the Witwatersrand, advantage is taken of them ; and, no doubt, this is true to a certain extent. Still, the point where I wish to support Mr. Thomas in his theory goes much further, and, to illustrate it, I refer you to the eastern portion of the South Wales coalfield, the Monmouthshire section, where the slips or lines of fracture are not observable, but nevertheless, are present and take a regular and defined course, and unless the manager finds out in which direction the lines of weakness run, and lays out his work accordingly, he is unable to work his coal efficiently. In examining your banket, the first thing I did when I got into a stope was to ask, "what is the right direction of the face in order that rock may be broken skilfully ?" The general answer I got was, "well, there is nothing regular. We get slips occasionally, and when we do we follow them, but, as a rule, they are not seen, and the stopes are worked the best way the miner thinks fit at the moment." That is fairly well proved by observing some of the stoping plans which are really very irregular, so far as the line of face is concerned. I followed this question of the line of weakness from one mine to the other, and not until I got to the Jumpers Deep did I find a man who had really taken the matter up seriously. When I went there Mr. Thomas mentioned it to me, and this was the first hope I had of finding someone to support my views and of putting this problem before other mine managers. I think to-day several managers are giving it considerable thought. Mr. Thomas' contribution should be carefully read and studied. I believe there are lines of weakness—although not observable, but well worth looking for—in the banket. If we can convince ourselves they are present, we ought to take advantage of them, and by so doing be able to increase our efficiency in breaking rock. That is all that I can say to-night. I presume the discussion on this paper will be continued. In the meantime, I will carefully read Mr. Thomas' contribution, and if there is anything I can add, I will be very glad to do so.

The President: We must thank Mr. Stuart Martin for his corroboration of Mr. Thomas's theories. The discussion of this paper is likely to lead us to more knowledge upon scientific mining. It would certainly appear that due advantage has not been taken of following up fracture planes hitherto, and this opens a somewhat new field for thought and instruction in mining work. We shall be glad if Mr. Stuart Martin can give us later on any further information on the subject, for I am sure it will be a matter of great interest and instruction to our mining members.

Mr. H. C. F. Bell (*Student*): Having read this paper with much interest, I should like to make a few remarks on one or two points, regarding blasting in the Rand mines.

I consider that the charging of holes is a very important process, and during the time I was hand stoping I paid particular attention to it. At first I used to put five or six sticks of gelignite or gelatine into a 4 ft. hole, charging in two at a time, and then rammed in about 8 in. of sand tamping; later on I put in the same number of sticks of explosive, but charged in only one at a time, and then put in about 30 in. of tamping. I only rammed in half-a-packet of tamping at a time, as a whole packet did not crush up in the hole but merely doubled up, leaving an air space at the middle of the packet. This change at once enabled me to increase the burdens on the holes by about 8 in.

One of the mine officials, who had been a miner all his life, told me that the extra amount of tamping made no difference, but I disagreed with him. He said that water was the best tamping to be had. I have had only a little experience with water tamping, but do not consider it nearly so good as well rammed sand. If the sand is fine and slightly damp, it is still better. Since the gases caused by the explosion of the dynamite tend to expand equally in all directions, the greater the resistance the tamping offers, the greater must be the pressure on the sides of the hole.

When I had a weak hole, in order to economise, I used to distribute the charge along the hole, by separating each stick of dynamite with 3 or 4 in. of tamping, at the same time I distributed the detonation by placing the primer in the centre of the charge; this prevented "bull-rings." The author's suggestion about giving strong hammer boys 5 lb. hammers is a good one, and I would suggest that better results might be obtained by making the hammer heads long and narrow, instead of short and thick as at present. This, I think, would cause a greater part of the momentum of the hammer to fall on the centre of the end of the jumper.

Mr. J. Moyle-Phillips (*Member*): I am glad to find the author advocating the distribution of charges over long holes. I have not seen his original paper on this theory, otherwise I should have communicated with him, as I have experimented along those lines myself. It was some time prior to the war that the idea occurred to me in consequence of my long holes having a tendency to "bull-ring." I was working on machines at the time, stoping. I started off with inserting 3 in. of tamping in the centre of the charge, and, as I fancied this gave better results, I split the charge up into three portions as follows:—(1) two cartridges; (2) 3 in. tamping; (3) one cartridge; (4) 3 in. tamping; (5) two cartridges; but soon found that at times the bottom portion of the charge remained unexploded, so I reverted to my first experiment, gradually increasing the length of tamping to 4 in. At that time I was working on the idea prevalent among our miners to-day, that as gelatine is the strongest explosive, so also it must be the quickest. After some time came the thought that if loose ground requires the quickest explosive, why is dynamite used in preference to gelatine? I then began to make enquiries concerning the manufacture and uses of high explosives, and found that the characteristic feature of dynamite *is its intense rapidity of explosion*, and that in big blasts, dynamite is often inserted along or among the charge to ensure its perfect detonation. Working along these lines, I found that a charge composed of four cartridges of gelatine and one of dynamite gave equal results to one composed of five cartridges of gelatine only. I then distributed my charge as follows:—(1) two gelatine; (2) 4 in. tamping; (3) one dynamite; (4) 3 in. to 4 in. tamping; (5) two gelatine; this method distributed my charge over from 3 ft. 11 in. to 4 ft. of the length of the hole, thus leaving only 2 ft. on top of the charge, and as I had a 3-ft. burden, I found that 18 in. of the hole was sometimes left, so eventually I distributed my charge as follows:—(1) two gelatine; (2) 4 in. tamping; (3) one dynamite; (4) 3 in. tamping; (5) one and a-half gelatine; (6) half dynamite for primer. This method gave me the best results, both as regards costs and quantity of rock broken. These experiments extended over a very interrupted period of over two years, and as no dynamite was issued to machine men, and what was obtainable was only in the form of the ordinary $\frac{7}{8}$ in. cartridge, I had to manufacture an 8 in. by $1\frac{1}{4}$ in. cartridge out of what I could beg, borrow, or steal. I think I only applied the mixing of the explosives to developing, about twice, and this I only accomplished after great argument and persuasion with a developer; he would not hear of my inserting tamping, and laughed the idea to scorn. I did not distribute the charges, but simply mixed the explosives by replacing a given quantity (weight) of gelatine by a like quantity of dynamite. The results I obtained were apparently equal to that of a full charge of gelatine, but with the advantage of being cheaper by the difference of the price between dynamite and gelatine. I have not been able to continue these experiments since the war, but to those who have opportunities I can conscientiously recommend a trial of the method. I am of the firm conviction that the reason why we do not get higher results from the charges we use is through imperfect detonation, and that the insertion of dynamite in the centre of the charge, and as a primer—replacing a like quantity of gelatine—will give the best results. The insertion of a primer and detonator in each portion of the charge will not overcome the problem of exploding the entire charge simultaneously unless exploded by electric fuses or "volley" fuses. I have experimented with the ordinary safety fuses and find that with three fuses cut of exactly the same length and fired simultaneously, no two of them will explode their detonator simultaneously, and this under ideal circumstances.

REMINISCENCES OF THE EARLY RAND.

(*Read at August Meeting, 1908*).

By M. H. Coombe (Member).

DISCUSSION.

Mr. J. S. MacArthur (*Honorary Member*): After a stormy voyage to Capetown, a pleasant trip through Cape Colony to Kimberley, and a wearisome coach journey to Johannesburg, I arrived there in April, 1890. Happily I forget much of the unpleasantness of the journey associated with the good old coaching times. I do remember, however, that a coach which would have been well filled with six passengers had to carry a load of twelve. We had to "sit tight" and be good natured—or be thrown out. The monotony of tight sitting was varied by an occasional walk where the dust was too deep for seven pair of horses to pull through, and when we came to a sluit the passengers jumped over and helped the horses. Fares were high, about £8 to £12 for Kimberley-Johannesburg stage. In modern days the train carries baggage at a less rate than the human passenger, but the coach charged more per pound for luggage than for humanity. The coach was the first-class conveyance, used principally by those who travelled at the expense of others. Those who travelled on their own account generally bargained to be conveyed with "urgent" goods by fast mule wagon for a modest fee of, say, £2, each passenger supplying himself with food, and co-operating with the transport riders in the toils and adventures of the trek. Probably these mule-wagon passengers had a pleasanter time than their coaching friends.

At that time Johannesburg was certainly the premier gold camp, but the glamour of Barberton had not quite faded, and Klerksdorp was also the centre of a gold industry, very promising if not very thriving. There were many who shook their heads wisely and said that Johannesburg was only a passing fashion, and time would show that Barberton was the place; others, with a knowing look and a grave whisper assured their acquaintances that they positively knew that Klerksdorp was the key to the situation.

I spent a few days at Klerksdorp and saw several batteries "just ready to start but"—there was no capital to develop the mine, or the mine manager had not arrived, or a barrel of lubricant had been lost on the way. There was always some good reason for the silence of the mill.

At Johannesburg some mills were going, but many were silent. The boom had passed and the blight had come. As the possibility and opportunity of speculation had dwindled, the crowd of irresponsibles who dubbed themselves brokers also dwindled till very few were left. The whole mining industry was in a precarious state, the rich surface patches had been exhausted, the average grade of ore was falling, working expenses remained high, and the dread of the blue ground under the oxidised zone deterred fresh capital and discouraged new enterprise. At least one-third of the houses and stores were unoccupied and gloom was general. Families were leaving, offices were being closed, cheap sales of pianos and safes blocked the Market Square.

There were many processes in the air—fine crushing by wonderful machines was one type of the infallible cures, amalgamation, where contact was induced by mechanical and electrical agencies, was another, chlorination with pressure or without was another, and some inventions included all these, so that whatever gold escaped one trap would surely fall into the next. Two or three years before this period the cyanide process had, with labour and pain, been evolved into existence in Glasgow under the brothers Forrest and myself, and was being brought into world-wide use by the Cassel Gold Extraction Co., who controlled the patents. Late in 1889 this company sent an expedition of a few skilled men, under the leadership of Mr. Alfred James, to erect a cyanide plant, for demonstrating the applicability and commercial value of the process in South Africa. This pioneer of the huge cyanide plants now in existence on the Rand was erected in the Salisbury battery, and many little troubles were avoided and difficulties smoothed by the manager —Mr. R. A. Michell—who was truly hospitable in his kindness to the little band of enthusiasts who bothered him with their cranks. The directors of the Cassel Co., recognising the importance of the ordeal, decided to send the ringleader of the inventors to take command, and thus came about my first visit to the Transvaal. By the month of June, 1890, the cyanide plant was ready for work. It was composite, available for agitation and for percolation. It had a vacuum attachment for hastening filtration, and provided everything but a filterpress, a piece of plant already tried and found wanting in its then design. It is a very different appliance now.

After a little preliminary work on our own account we prepared for a demonstration of the process and consignment; on the challenge of a celebrated metallurgist, now almost forgotten, who jeered at the bare idea of cyanide being a commercially useful gold extractor, we decided to work on the Percy concentrates. These were ordinary underground pyritic concentrates containing 20 oz. gold per ton.

The representatives of the gold industry were invited to check us at every point, and they came prepared to be very exacting, but when afternoon became evening and evening became night, refreshment, recreation and rest became necessary, so with most of them watching became whist and whist became sleep. One of them, however, Mr. Hennen Jennings, was not to be tempted; he attached himself to me like an old friend, and I am glad to say he has not shaken me off yet. He took nothing for granted, measured every vat and every pipe, sampled the water, the lime, the cyanide and the zinc to make sure there was no humbug. For two days and two nights the trial lasted until late on the third day, when a small ingot of gold was turned out of the little kerosene smelting furnace. The industry is not now called on to deal with rich concentrates, in fact, the concentrates are not much in evidence since cyanide became dominant and this Percy lot was most exceptional; so in deference to the wishes of my colleague and other technical friends we used a 2% solution, though in our ordinary work in the laboratory we used solutions as weak as those now in general use. I must admit that my friends had good reason for their wish to use a comparatively strong solution, for cyanide was not in these days the pure salt of reliable composition that it is now. In the solid, its true cyanide contents were erratic; in solution, it was without sufficient reason, considered uncertain and cranky, and, after all, they said that the extra strength would not hurt if it did not help. Probably Mr. W. A. Caldecott, who was our first South African assistant, remembers many of the incidents of this period; his scientific acumen was then as now—an invaluable guide in a new field of industry. The demonstration was over, the gold extracted to the extent of 98%, but the industry was not convinced. Those who knew most were perhaps incredulous but not contemptuous; those who knew least had to be contemptuous to maintain their dignity. One gentleman, who had been an expert telegraph operator and knew all about batteries, "chemicals and things," condemned it roundly as the result of his accumulated wisdom. Once, coming as an unknown passenger in the Pretoria coach, I heard the process discussed as "bound to be a failure," "all about it known long ago," "positively ridiculous," and myself as "an honest sort of fellow but knows nothing," and "all right in the laboratory but no good outside it." The temptation was too great, and to the consternation of the critics I disclosed myself in the last mile.

In these days the professions of journalist, mining engineer and metallurgist were frequently combined in the one person—generally it was an unhappy coalition. One of the gentlemen was shown over the works by me, everything was fully explained and understood, at least so I flattered myself, but as he was about to go he said, "now, my dear fellow, you really cannot fool me as you fooled the others, what is it you have up your sleeve?" I have always regretted the loss of this magnificent opportunity for telling some wonderful fairy tale. Another type of man whom I dreaded was the "practical man," who revelled in shirt sleeves turned up to the elbows and concealed his ignorance in a fluent jargon of weird science and torrid profanity. Fortunately he is now relegated to his proper sphere and bosses the Kafirs, and well trained scientific young fellows have taken his place and discharge responsibilities the "practical man" never dreamt of. A type of man whom I admired then was the clerk, stockbroker, sugar-planter or storeman who, by force of circumstances, found himself in charge of a mine. Many of these men, who started by knowing little or nothing of mining and metallurgy, quietly went on learning and working. In the end most of them had to give their place to better trained men, but they served their day and did their share in founding the industry.

By 1891 the African Gold Recovery Syndicate, into whose hands the control of the MacArthur-Forrest patents have passed, had made some profit from the treatment of 10,000 tons of Robinson tailings and had arranged for the treatment of a similar parcel of tailings at Fever Creek. As profits had actually been made, leaders of the industry were convinced that the process was good and, as the process was at the booming stage, I again visited the Rand in 1891. Everything was changed, every house was occupied, every store was filled, every merchant was busy, there was a distinct feeling of confidence in the future, fresh enterprises had been taken up and were being initiated every day and the Rand industry, as it stands now, was clearly foreshadowed. Of course, wise men were at it again, and each wiseacre took credit to himself, saying he knew from the very beginning that there was something in that MacArthur-Forrest patent, and each one took credit for the encouragement that he had given to the inventor, and told all and sundry that but for him the process would have been given up in despair.

By the introduction of cyanide the gold industry everywhere, but particularly on the Rand, became really an industry—which science, combined with good business management, has changed from a mining speculation into a safe investment.

I was again on the Rand in 1895-6, at the time of the Jameson Raid. My occupation was then the defence of the patents which protected the process which, next to the existence of the main reef series, was the most important factor in the development of the wealth of the South African Republic. Unfortunately, since that time I have had no interest calling me to the Transvaal, but I still regard Johannesburg and the many friends that I met there with affection, and the episodes of that period form a series of pictures in my mind, which I trust may never fade.

SMALL MINES OF RHODESIA.

(*Read at September Meeting, 1908.*)

By B. I. Collings (Member).

DISCUSSION.

Mr. Wilfrid Taylor *(Member)*: The first question that occurs to one in reading a paper on this subject is; do the small mines pay? Are a fair percentage of the men who risk their money and forego the comforts of civilization making fortunes, or even a decent living? That a few mines have paid handsomely for some months probably goes without saying. A mine for instance that will yield a £1,000 worth of gold per month, crushing with two stamps, should certainly show a handsome profit on working costs, but one is impelled to ask, how much time and money did the owner of such a mine put in before making his rich strike, and did he have a sufficient number of outputs to pay him for his preliminary work. Of course he probably did not, but he will be referred to as one of the men who are making a net profit of £800 a month, and held up as a bait to induce others to try their luck. If one is successful, there is a moderate prize to take out of the country, or bury in the ground in the search for further "rich shoots." If unsuccessful, one has lived a free and untrammelled life in the open, and gained by a widened experience. These remarks would appear to apply to the company worked mines also, as although there is a steady increase in the output and reduction in working costs, there is consistent conservatism in the matter of dividend distribution.

Touching some of the points referred to in the author's paper: the disappearance of gold and sometimes of reef at shallow depths is a disappointing feature, especially as the ancients worked most of the rich shoots down to water level. I have, however, information from one mine, that where the reef broke up and values fell to practically nothing at 120 ft., it was located again by boring at considerable depth, and of high grade. This would suggest a surface disturbance after impregnation.

I have known similar cases to case ii. (pp. 78), mentioned by the author. It should be understood that sinking in the filled up workings of the ancients is laborious, dangerous, and expensive in timber, consequently shafts are usually put down in the strips of ground left between the old excavations, driving commencing at or a little below water level, or where good values are found. Rising a comparatively short distance from the shaft should then give a good idea of the backs, and it has to be assumed that the measurement found will hold throughout the length of reef.

It would be interesting to hear the present price of coal to the mining centres. When I was in Rhodesia, some years since, the cost of wood fuel was about 24s. per cord, including 7s. 6d. paid to the owner of the farm. Taking two cords of wood as equalling one ton of coal, gives the coal and railway companies anything up to 48s. per ton to work on.

Following the value of a reef by panning is liable to lead to unfortunate errors. One case where the pan showed excellent results, fire assay proved 25% to 50% silver in the gold. Another case, a prospector under the impression he had discovered something really valuable, was intensely disappointed on finding his "tail" in the pan to be chromate of lead. In still another case, a very rich show in the pan disappeared on the addition of nitric acid.

It is interesting to note the author's conclusions on the relative value of the various methods of crushing rock. No doubt every one would prefer the gravity stamp if it were not for its unportability and first cost. It can certainly stand more rough handling than most other machines, and the small worker is not usually a scientific worker, but rather a man of untiring energy, full of resource and a believer in luck.

If, as the report of the Rhodesian Chamber of Mines says, he proves the permanence and value of the reefs in the country, capital will be induced to extend operations. If on the other hand, the reefs pinch out, or are too poor for profitable working, Rhodesia will develop into a first-class farming country, and the small mines will have served their purpose in providing a hard working population.

ZINC DUST PRECIPITATION AT CERRO PRIETO.

(*Read at September Meeting, 1908.*)

By Robert Linton (Associate).

DISCUSSION.

Mr. T. Lane Carter (*Vice-President*): In connection with this paper from Mexico, it might be of interest to have a brief note from the Homestake, South Dakota, U.S.A., cyanide plant, where the zinc dust method of precipitation is used with great success.

One of the most striking differences between the enormous cyanide plant of the Homestake Gold Mine and those on the Rand, is the method used for precipitating the gold from the solutions. On entering the plant, one looks in vain for the large boxes filled with zinc shavings we are accustomed to see in the extractor houses of the Rand. In their place he finds large wooden vats into which the gold solution flows. As a rule the solution runs by gravity from the vat to the filter press. Zinc dust (the zinc dust used is, by the way, cheaper than shavings), is added to the tank automatically as the solution flows to the filter press. The precipitation of the gold by the zinc dust, and the retention of the precious material by the filter press is almost perfect. One remarkable fact is the rapidity with which the zinc dust does its work. In one press I saw, the point at which the zinc entered the solution was only a few feet from the press, and yet in this short distance, the zinc dust completely precipitated the gold.

This zinc dust idea is no new one. It has been up before our society before, namely when Mr. Packard gave a paper on the subject (see vol. ii., p. 720). The discussion which followed the paper was distinctly adverse to the scheme. In many places of the world where it was tried, the zinc dust idea was a failure, partly through ignorance, but principally through a lack of "stick-at-edness." Precipitation by zinc dust is a distinct success at the Homestake mine, Mr. A. J. Clark, manager of the cyanide works, kindly consented to contribute some interesting figures on the results he has attained, and from these it will be seen how well zinc dust precipitation shows up in this, the largest cyanide plant in the world. The figures I think are impressive, and I believe the zinc dust scheme could be taken up with advantage in South Africa. Of course, I do not mean to say that our precipitation boxes should be thrown out in the old plants and the zinc dust scheme introduced. but in new plants it might be well worth considering some of the advantages of the zinc dust precipitation. Besides its cheapness and cleanliness, one of the great advantages over the precipitation boxes is that the danger of theft is greatly reduced. The precious metal is caught in a large filter press, which is securely locked. The gold is cleaned up only once a month, and this is not the laborious operation it is with us. When the press is opened up, the gold slime is taken direct to the briquette plant, where the slime is made ready for smelting, with the minimum amount of handling. One clean up per month, as against three is an advantage. Another point is that the overlap of the cyanide gold from one month to another is reduced to nil. The cyanide clean-up at the end of the month is as complete as in the mill. The differences between monthly theoretical and actual extractions are therefore reduced to a minimum, and you do not get the violent monthly fluctuations we get on the Rand. In using the zinc dust method, care must be used in getting the right kind of zinc. At the Homestake, they specify that the zinc dust carry at least 2% lead. A little nitrate of lead is also used. Another important point is to keep the press air tight, so as to prevent oxidation. It is a pleasure to testify to the good work accomplished at the Homestake plant by the zinc dust method. Their success proves that as good results can be obtained as the precipitation by zinc shavings that we are accustomed to on the Rand.

The President: Before closing the meeting I wish to extend to you all my heartiest greetings for the festive season which is approaching. I trust that all our members will spend as happy a time as possible, and that the coming year will add to the prosperity, satisfaction, and comfort of you all.

Obituary.

It is with great regret that the death is recorded of Mr. JAMES LAURIE HAMILTON (Member), who died from the effects of a revolver accident on the 23rd December, 1908.

Mr. Hamilton, who was a son of Mr. William Laurie Hamilton, the well-known mining engineer on these fields, joined the Society in 1903, and had occupied responsible positions with the Rand Mines, Ltd. At the time of his death he was the Acting Manager of the Ferreira Deep, Ltd. The funeral at Braamfontein on Xmas Eve was largely attended.

Contributions and Correspondence.

"MISLEADING ANALYSES OF AIR"—A CORRECTION.

In our March number* I published a letter under the above heading—which I have now to withdraw, as I find that I have myself made a most serious error in course of writing it. I entirely overlooked the fact that the quantity of water vapour diminishes during the analysis, being absorbed by the reagents in proportion as the gas-space contracts. In consequence of this, the Orsat apparatus automatically corrects the supposed error I wrote about. Thus, although the sample of fresh air there mentioned consists, when inside the burette, of $CO_2 = 0·05\%$, $O_2 = 20·27\%$, $N_2 = 76·45\%$, and $H_2O = 3·23\%$, yet on analysis the 3·23% of water partially disappears because there is at the end only the space occupied by the nitrogen to hold it, viz., 79% of the original space; hence 21% of 3·23% also disappears and is counted as oxygen or 0·68% on the original; and this raises the oxygen from 20·27% to 20·95%, which is exactly the percentage existing in the dry air before it entered the apparatus. I had thus got hold of a "mare's nest," and have to apologise to the Society for having been so careless. The statement about deficiency of oxygen in mine air is also wrong, for, although there is less than 20·5% of oxygen *by volume* in the actual moist air below, yet on analysis the figure would again be raised to 21%.

I have found out the true cause of the error of about 0·4% in oxygen which one gets with this make of apparatus. It is due to a shifting of the apparent position of the chief meniscus according as the reading is taken near the bottom or farther up the burette. Most of these Orsat burettes give a false reading of the zero point when illuminated from above, and in using them the different menisci have to be most carefully distinguished.

I have to thank Dr. Haldane for calling my attention to my error—which was also noticed here by Dr. Weiskopf.

JAMES MOIR.

BATTERY AND CYANIDE GOLD SMELTING.

I regret to have to call attention to a mistake made by Mr. A. Thomas in replying to the criticism on his recent paper on gold smelting.

Referring to my contribution, he credits me with having stated that I would collect the gold in lead, as in Tavener's method. This, I need hardly say, is not the case, as the method I indicated recovers high grade bullion from zinc slime in one operation only.

G. H. STANLEY.

MILL PULP TABLE.

By C. O. SCHMITT (*Member*).

1 cub. ft. solid ore = 165 lb. 1 cub. ft. water = 62·5 lb.

Ratio by Weight Water used to Ore Crushed.	Weight of Pulp per cub. ft. lbs.	STAMP DUTY PER 24 HOURS.								
		5·5 tons.	6 tons.	6·5 tons.	7 tons.	7·5 tons.	8 tons.	9 tons.	9·5 tons.	10 tons.
		QUANTITY OF PULP PER STAMP PER MINUTE IN CUB. FT.								
6 to 1	68·5893	·7796	·8505	·9213	·9922	1·0631	1·1340	1·2757	1·3466	1·4175
7 ,, 1	67·7658	·9018	·9838	1·0658	1·1477	1·2297	1·3117	1·4757	1·5577	1·6397
8 ,, 1	67·1322	1·0241	1·1171	1·2102	1·3033	1·3964	1·4895	1·6757	1·7688	1·8619
9 ,, 1	66·6394	1·1463	1·2505	1·3547	1·4589	1·5631	1·6673	1·8757	1·9799	2·0841
10 ,, 1	66·2419	1·2685	1·3838	1·4991	1·6144	1·7297	1·8451	2·0757	2·1910	2·3063
11 ,, 1	65·9141	1·3907	1·5171	1·6436	1·7700	1·8964	2·0228	2·2757	2·4021	2·5286
12 ,, 1	65·6349	1·5130	1 6505	1·7880	1·9255	2·0631	2·2006	2·4757	2 6132	2·7508

---

* See this *Journal*, vol. viii., March, 1908, p. 280.

Proceedings
AT
Special General Meeting, December 19, 1908.

A Special General Meeting of members was held in the Council Chamber of the Transvaal Chamber of Mines, Johannesburg, on Saturday evening, the 19th December, 1908.

The attendance was the same as at the Ordinary General Meeting, Mr. R. G. Bevington (President) in the chair.

The President stated that the meeting had been called for the purpose of considering and, if approved, adopting, with or without amendment, the following proposed additions to the Constitution and Rules of the Society, regarding Local Sections which had been published in the last two Agenda papers.

It was resolved to take the clauses as read :—

1. A local section may be formed by the Council whenever at least twenty members or associates residing in or identified in interest with any locality, district, colony or country, whether in South Africa or not, desire to have local facilities for meetings, and sign and forward a requisition to the Council, asking to be permitted to form a section for that purpose. The consent of the Council being obtained, the signatories shall elect a local committee and officers and formulate Bye-laws, subject to the approval of the Council of the Society, for conducting the business of the Section.

2. The Chairman of the local committee shall *ex officio* become an Hon. Vice-President of the Society, and the Hon. Secretary shall *ex officio* become a Corresponding Member of Council, during their respective terms of office.

3. Places and dates of Sectional Meetings and the business to be transacted thereat shall be arranged by the local committee of the Section.

4. The Sectional subscription, if any, for defraying local expenses shall be fixed by the local committee, but shall not exceed 10s. 6d. per annum for each member or associate.

5. Members and associates resident in or identified in interest with the district included in the Section shall, upon application to the Hon. Secretary, be entered in the register of the Section and be entitled to the privileges thereof.

The local Committee and all officials of the Section shall be elected annually by the members on the register of the Section.

6. Papers read before any Section and the discussions thereon shall, subject to the approval of and editing by the Editorial Committee of the Society, be printed in the Journal of the Society under the heading of the Section.

7. The expense incurred in the publication in the Journal, of Sectional papers and contributions to discussion shall be borne by the general funds of the Society, and a contribution to the local expenses of the Section may be made from the general funds of the Society by the Council.

8. The Society as a body shall not be responsible for any local expenses or obligations incurred by sections, although it has discretion to contribute thereto.

9. Amendments in, alterations of, or additions to the bye-laws of a Local Section shall be submitted to the Council, and shall not become operative or of any effect unless and until they have been allowed by the Council, and notice of such allowance has been given by the Secretary.

10. The Council may, at a special meeting called for that purpose, declare by resolution that a Local Section shall be suspended in operation, or shall be dissolved :—

(A) If it shall appear that the further continuance of the Section is not in accord with or conducive to the interests of the Society, or the furtherance of its objects.

(B) Whenever three-fourths of the Members of the Local Section vote in favour of a resolution, proposed at a meeting called for the express purpose, that it is desirable such Section shall be suspended or dissolved.

Mr. W. A. Caldecott (*Past-President*): These proposed additions to our Constitution speak for themselves. They are largely based on the rules of the Society of Chemical Industry of London, which has been so conspicuously successful in founding active sections in various parts of the world—Canada, Australia and the United States. This Society has frequently served as a model for us in the past, not only in the form of its publications, but what is far more important in the example set us of that liberal spirit, free from any tendency to exclusiveness, which has benefited us so greatly. For the wording of the proposed regulations we are indebted to the courtesy of of our Honorary legal advisers, Messrs. Van Hulsteyn, Feltham and Fry, who have always so readily assisted us in legal matters.

As we may fairly claim to be the premier Gold Society in the world, the Council hopes that the invitation that is now extended, and the offer of financial assistance for the purpose therein made, to the hundreds of our members away from this centre will facilitate the formation of local sections wherever practicable, both for the especial benefit of members in any district, and for that of the Society as a whole.

I beg formally to move that these proposed additions be included in the Society's Constitution and Rules.

The motion was seconded by Mr. C. B. Kingston and unanimously agreed to.

The meeting then closed.

Visit to Premier Diamond Mine.

On Saturday, Dec. 12th, about 100 members of the Society visited the Premier Diamond mine (by kind permission of the directors), which is situated almost due east of Pretoria, just off the Delagoa Bay railway line, and about 76 miles from Johannesburg.

After a thorough inspection of the works the party lunched at the Premier Hotel. Mr.

Bevington (President of the Society), in a few apt sentences, expressed the thanks of members to the board of management for their courtesy. Mr. McHardy, junr., in the absence of his father, the general manager, owing to indisposition, in the course of a reply, said that the management was always ready to consider any ideas which might be made to improve working methods.

The party returned to Johannesburg abont 5 p.m. by special train after a most enjoyable visit.

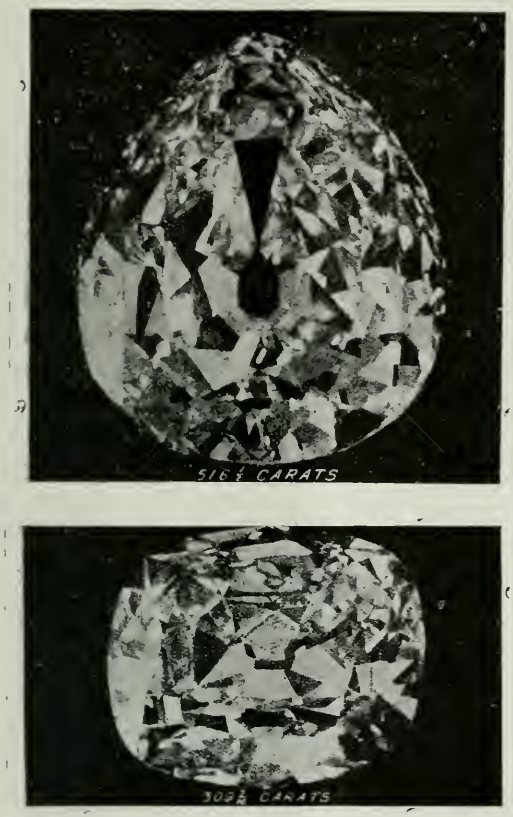
The following is a description of the mine and plant:—The Premier pipe is bounded by hills of an average height of 100 ft., principally of sandstone and quartzite, and these form the rim rock of the pipe. The area of the mine is equal to 3,570 claims of 900 Cape square feet. In English measurement this represents 3,561,300 square feet, or, say, 78 acres. The mine is being worked in levels or terraces, 50 ft. in depth, and, in view of the area of the pipe and the hard nature of the rim rock, it is officially considered that it will be possible to continue the open workings to a depth of 1,200 or 1,500 ft., which will represent approximately 500,000,000 loads of 16 cub. ft., or, say, 40 years' work, calculated on an output of 12,000,000 loads per annum. Nine boreholes have been put down within the pipe area to depths varying from 300 to 1,001 ft., and the diamonds extracted from the core obtained show a favourable yield throughout.

It having been the policy of the company to start operations without delay, two temporary washing gears were erected, comprising respectively three and eight pans each. Work was commenced with the former in April, 1903, and with the latter in January, 1904.

In view of the facilities afforded for working on a large scale, however, arrangements were subsequently made for the erection of an up-to-date direct double treatment plant capable of washing 20,000 loads of 16 cub. ft. per day of 24 hours, and work with this gear, which is the largest of its kind in the world, was commenced in November, 1905. Up to date the three gears have treated 24,250,000 loads, from which 6,750,000 carats of diamonds were recovered, equal to a ton and a quarter. Amongst many other large stones, the mine has yielded the largest known diamond in the world. This stone—the "Cullinan"—is of the best quality, and, when found, weighed 3,025¾ carats.

Method of Treatment.—The following is a description of the method of washing the ground by what is known as the direct double-treatment system:—The ground is hauled up from the mine by means of a mechanical haulage to the apex of the incline, at which point the trucks disengage from the wire and gravitate along the bridge, being tipped automatically as they pass each of the grizzleys, or receiving hoppers. After passing the last grizzley the trucks are "righted" automatically and re-engage on to the return wire to the mine. When the ground is tipped into the grizzleys, the bars of which are set 2½ in. apart, the finer ground passes between these bars through two sets of 4-ft. corrugated rolls, and is joined by the coarse ground after it has been reduced by the gyratory crushers. The 4-ft. rolls reduce the ground to ⅞ in., and thence it falls to the elevator pit. From this point the whole of the ground is elevated and discharged into a four-way hopper for distribution to the top pans. In the pans the

Photograph of the Cullinan Diamond before being Cut.



The Cullinan Diamond after Cutting and Polishing.

ground is carried round by means of revolving arms, fitted with triangular teeth set in the form of a spiral. These teeth force the diamonds and other heavy deposit with which the diamonds are associated to the outside edge of the pan, the lighter material flowing out at the discharge situated on the inner rim. The discharge from the top pans passes through a set of 6-ft. smooth rolls, which finally reduce the ground to $\frac{3}{16}$ in., and thence into the bottom pans, the overflow from the bottom pans passing to the tailings elevators, which discharge the waste ground in the valleys below the machine.

The Grease Tables.—Arrangements are made to extract the heavy deposit from the pans continuously, one load of deposit being obtained for every hundred loads of ground treated. This deposit is conveyed to the pulsators, where it is classified and gravitated, about 75% of the lighter material being discarded, and the remainder, which comprises the heaviest of the deposit, with the diamonds, passes over the grease tables. The diamonds, together with a very small proportion of concentrates, adhere to the grease plates, the remainder of the concentrates passing over the plates to the "dump." The plates are cleaned at intervals, and the deposit separated from the grease by means of a boiling process. The deposit is then placed on the sorting-tables, where the diamonds are picked out by hand.

As the grease tables do not take up all the diamonds, the concentrates are passed through a tube mill, the resultant pulsated and sorted by hand. Only a very small proportion of the total recovery is secured from this final operation. This additional process has been in operation close upon a year, and it will be observed that the tube mill has been pressed into service, so as to obtain the greatest recovery of diamonds as it has been, in manner of speaking, to extract the last particle of gold, on the Rand. In one of the houses was to be seen a short tube mill of new type, the main feature being that it runs on wheel supports instead of on the ordinary journals.

Steam and Electric Plant.—The steam-power plant comprises two vertical triple expansion engines, each capable of developing 1,500 h.p. One is kept as a standby. Shafting of the main gear is run at 200 revolutions per minute, which is considerably faster than Rand battery practice; but, of course, the circumstances are dissimilar. The electric generating plant consists of two 340 kilo-volt-ampere three-phase alternators, which generate current at a pressure of 5,000 to 5,300 volts.

The lighting plant comprises two 125 kw. direct current dynamos, which generate current at 220 to 250 volts.

A battery of Babcock and Wilcox boilers, which are hand-fired, equal to 5,500 h.p., supply steam for driving the entire plant. The coal bunkers adjoining the boilers have a capacity of 800 tons; the coal is delivered direct from the railway trucks into the bunkers. Three overhead travelling cranes, operated by electricity, control the whole gear and facilitate the work of renewals and repairs.

Water Supply.—In order to make provision for a permanent water supply, a pumping station was erected at the Wilge river, about 23 miles distant from the mine. A concrete weir 900 ft. long and 30 ft. high has been constructed across the Wilge river, forming a reservoir having a capacity of 1,100,000,000 gallons, which is full at the present time. The catchment area is equal to 1,200 square miles. Four triple expansion engines working ram pumps have been erected, each pump being capable of delivering 1,000,000 gallons per day. At present 3,000,000 gallons per day are being pumped to the mine at a cost of about 2d. per 1,000 gallons. Steel pipes 20 in. and 21 in. in diameter respectively have been laid from the pumping station to the mine.

Additional Plant.—A new washing plant, styled No. 4, is now in course of erection adjoining the No. 3 gear, and when completed it will be capable of treating 25,000 loads per day, which will enable the output to be increased to 45,000 loads per day of 24 hours. The whole of the washing plant will then be concentrated at the one point 200 ft. above the level of the surrounding country, thus affording unique facilities for the disposal of tailings to the extent of 1,060,000,000 loads, assuming the tailings to be carried out on the same level as the gear.

The No. 4 gear consists of seven units, two of which should be ready for work in a couple of months' time. It may then be decided to shut down the smaller gears in order to concentrate work, and thus reduce working expenditure.

Considerable interest was taken in the foundry started some 12 months ago: besides economising time, it enables scrap-iron to be used up. A pattern of a 9-ft. scraper wheel was noticed. The foundry is equipped with a 4-ft. cupola.

There are at present 8,000 natives and 800 white men employed on the company's works. The total white population is over 2,000. Much interest was taken in the visit to the compounds and hospitals, in which the native workers are excellently looked after.

SUMMARY.

Extent of Mine.—Average length : $\frac{1}{2}$ mile (greatest length is actually 2,900 ft.)

Average width : $\frac{1}{4}$ mile.

Thus area ($\frac{1}{2}$ by $\frac{1}{4}$ mile) : $\frac{1}{8}$ by 640, equals 80 acres (actually 78·68 acres).

Claim area : 3,570 claims (30 by 30 Cape feet).

Loads.—One foot deep represents over whole pipe area 343,000 loads (16 cub. ft. in solid).

One level (50 ft.) represents over whole pipe area 17,000,000 loads

One year's working on basis of 12,000,000 loads per annum, equals a depth of 35 ft. over whole pipe area.

Open Workings.—Assuming each level 50 ft. and grade 1 in 8, the horizontal distances between onsetting points would be 400 ft. There would thus be seven levels, the bottom one being at the average vertical depth of 350 ft., or 2,800 ft. from the northern rim of pipe. The number of loads of ground above this 350-ft. level would be 120,000,000, or, on a basis of 12,000,000 loads per annum, would represent 10 years' work.

Levels.—Height above sea level, centre of pipe, 4,800 ft. Fall from south rim to north rim of pipe is 90 ft. Foundation of pump at Wilge river is 4,618 ft. above sea level. Highest point on pipe line is 5,122 ft. above sea level. Reservoir at mine is 5,000 ft. above sea level. Cullinan station is 4,923 ft. above sea level. Length of pipe line from Wilge River Waterworks is $22\frac{3}{4}$ miles. Capacity of reservoir at mine is 10,000,000 gallons. Capacity of reservoir at Wilge river is 1,100,000,000 gallons. Catchment area of the Wilge river water supply is 1,200 square miles.

Gears.—No. 1 : Three pans—two of 14 ft. diameter and one of 12 ft ; engine, 64 h.p. No. 2 : Eight pans of 14 ft diameter ; engine, 85 h.p. No. 3 : Forty pans of 14 ft. diameter, in five units, each comprising—two "Comet" crushers, one set double 4 ft. rolls, one feeding elevator, eight 14-ft. pans, two sets of 6 ft. smooth rolls.

Horse powers : 225 h.p. per unit, equals 1,125 h.p. ; 250 h.p. for main haulage, 50 h.p. for pulsators, 33 h.p. for conveyors, 160 h.p. for pumps.

Number of loads hauled, about 27,000 per day of 24 hours

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

DETECTION OF TELLURIUM.—"The most reliable method to detect even small traces of tellurium in large quantities of pyrites is as follows, observes *L'Elettricista* of July 15. The pyrites are completely oxidised with strong nitric acid, then evaporated to dryness and absorbed with strong hydrochloric acid. Finally it is boiled, diluted with water, and the insoluble residue filtered. During all the time of filtration and boiling a rapid jet of sulphur dioxide is run through until the filtrate is completely saturated with gas. The tellurium and gold will all be precipitated in the metallic form. These two metals are then filtered and washed with water until freed from the salts from filtration. To remove the tellurium from a paper filter it must be washed in an evaporating dish. When evaporation has attained the proper degree, strong sulphuric acid is added, and gradually heated. When tellurium is present a violet-pink colour appears quickly."—*Mining Journal* Sept. 12, 1908. p. 331. (G. H. S.)

THE CORROSION OF IRON.—The experiments conducted by Heyn and Bauer at the Royal (Prussian) Institution for Testing Materials, at Charlottenburg (*Stahl und Eisen*, Oct. 28), show that the corrosion of iron may proceed when carbonic acid is completely excluded, and that even when the air contains 15% of this acid the amount of corrosion is only twice that produced by pure air ; consequently, the small percentage of carbonic acid in the atmosphere has practically no effect on the corrosion of iron. On the other hand, the rusting of iron in or under water is greatly facilitated when the supply of dissolved oxygen is replenished by the passage of a current of air through the water ; and the higher the partial pressure of the super-incumbent air (or oxygen) the greater the corrosion. Pure carbonic acid, in the absence of oxygen, merely dissolves the iron, with liberation of hydrogen. As a preventive measure, the suspension of bags of charcoal in the water has been found to lessen the corrosion by 75 to 80% ; and this method seems to promise well for practical application. Copper and nickel in contact with the iron increase its liability to corrosion, whereas cast-iron lessens the corrosion of ingot iron by 28 to 50%, and a similar effect is obtained by placing iron that has been overheated in contact with ordinary iron. Taken separately, the rate of corrosion of ingot iron, welded iron, and cast iron in water is about the same; but in presence of dilute (1%) sulphuric acid the ratio is 1, 2, 100. Amongst other features, the experiments reveal the error of the general opinion that solutions of alkali carbonates are able (unconditionally) to protect iron from rusting, and that the alkali chlorides and sulphates accelerate corrosion, all the solutions of these last-named salts having in reality a smaller corrosive action than distilled water, owing to their diminished capacity for absorbing oxygen from the air. Chromic acid and its potash salts afford the best means of preventing rust. Cement that is in course of setting deprives the water with which it is mixed of the power of rusting iron; but hardened cement has not this property."—*London Mining Journal*, Nov. 7, 1908, p. 577. (A. R.)

DETERMINING SULPHO- AND FERROCYANIDES, ETC. —"The following tests have been elaborated from a study of the working cyanide solutions from several cyanide plants in Mexico treating gold and silver ores, which also contain more or less copper

The following solutions are required:

STANDARD SOLUTIONS.

Silver nitrate—13·05 grm. $AgNO_3$ in 1 litre of water.

Mercuric chloride—20·846 grm. $HgCl_2$ in litre.

N/10 permanganate—3·18 grm. $K_2Mn_2O_8$ in 1 litre.

APPROXIMATE SOLUTIONS.

Sulphocyanide to approximately equal permanganate.

Zinc sulphate solution. 10% $ZnSO_4.7H_2O$.

N/10 ferrocyanide—42·2 grm. $K_4FeCy_6.3H_2O$ per litre.

(1) Take 10 c.c. of solution. Add a small crystal of potassium iodide and a little soda solution and titrate with silver nitrate till there is a permanent yellowish cloudiness

Result = 'a' c.c.

This result represents potassium cyanide equivalent to the alkaline cyanide + the zinc potassium cyanide.

(2) Dissolve a small crystal of potassium iodide in a few c.c. of water, and add 5 c c ferrocyanide solution and a pinch of bicarbonate of soda. To this add mercuric chloride solution drop by drop, till there is just a permanent red precipitate. (A convenient amount of iodide is enough to combine with from 1 to 2 c.c. of the mercuric chloride solution.) To this add 10 c.c. of the cyanide solution to be tested, and then titrate with mercuric chloride until there is a permanent reddish colouration.

Result = 'b' c.c.

This result represents potassium cyanide equivaltent to the alkaline cyanide + the zinc potassium cyanide + the cuprous potassium cyanide + $\frac{3}{4}$ of the cupric potassium cyanide.

Note.—If silver be present there will be a lemon yellow colouration before the end point, which will then appear more or less of a deep orange colour. If silver be present in large amount, the amount of cyanide combined with it must be calculated and deducted from the amount shown in the test above to get the true value of 'b.'

(3) Take 10 c.c. of the cyanide solution, add 20 c.c. sulphocyanide solution and 5 c.c. zinc sulphate solution. Titrate with mercuric chloride solution till the precipitate remains of a permanent faint yellow tinge. This is best seen by placing the vessel in which the titration is performed on a sheet of clean white paper.

Result = 'c' c.c.

This result represents potassium cyanide equivalent to the alkaline cyanide + the zinc potassium cyanide + $\frac{1}{2}$ of the cupric potassium cyanide.

(4) Acidify about 10 c.c. of water very strongly with sulphuric acid and cool. Then add 10 c.c. permanganate and titrate this with the cyanide solution. In the presence of ferro and sulpho-cyanides, at the end of the reaction when the pink colour of the permanganate is removed, the solution will remain of a brownish or greenish yellow colour. Calculate the reducing power of 10 c.c. of solution in terms of permanganate.

Let 10 c.c. of solution reduce = 'd' c.c. permanganate.

This result gives the amount of permanganate reduced by the sulphocyanide + the ferrocyanide + the whole of the copper.

Note.—Sulphides and organic matter are not considered to be present. Should they be present, they must either be removed previous to testing, or estimated and allowed for.

(5) To 25 c.c. of solution add 5 c.c. ferrocyanide solution and acidify with H_2SO_4; then add 5 c.c. zinc sulphate solution and make up to 50 c.c. Shake up and filter through a dry filter. Titrate 20 c.c. of the filtrate with permanganate.

Result = 'e' c.c.

This result gives the amount of permanganate reduced by the sulphocyanide alone. Then—

Alkaline + zinc double cyanide, estimated as $\text{KCy} = \frac{a}{10}\%$

Cupric double cyanide $= \frac{c - a}{5}\%$

Cuprous double cyanide $= \frac{a + 2b - 3c}{20}\%$

Potassium sulphocyanide (KCyS) $= e \times {\cdot}0162\%$

Potassium ferrocyanide $K_4FeCy_6.3H_2O =$

$$d - \frac{2b - a - c}{2{\cdot}6} - e \times {\cdot}422\%$$

Total copper $= \frac{2b - a - e}{40{\cdot}9}\%$.

In the tests given the results have all been determined on 10 c.c. of solution. If convenient, a larger amount may, of course, be tested, the quantities of reagents added being increased and the factors for the results being decreased proportionally."—LEONARD M. GREEN.—*Institute of Mining and Metallurgy*, *Bulletin No. 49*, Oct. 1908, p. 6. (H. A. W.)

LEAD IN ORES.—"Technical Method. Low (*J. Am. Chem. Soc.*, XXX., 587). The Pb is obtained in the form of sulphate by the ordinary method (treating with HNO_3, etc., and finally evaporating with H_2SO_4). The $PbSO_4$ is dissolved out from the SiO_2, etc., by means of hot solution of $NaC_2H_3O_2$ containing a little free acetic acid. From this solution it is precipitated by $K_2Cr_2O_7$ (on 0·5 gm. of ore use 10 c.c. of a 5% solution). Boil for a few minutes, filter, wash once only, then open up the filter, and rinse the $PbCrO_4$ with a hot solution of $H_2C_2O_4$ into the flask, add alcohol, and boil until all the CrO_3 has been reduced, and the precipitate converted to PbC_2O_4. Filter, and wash 10 times with cold water, treat paper and all in a flask with hot dilute H_2SO_4, and titrate with standard permanganate.

The acetate solution for dissolving consists of a cold saturated solution diluted with an equal bulk of water and 40 c.c. of an 80% acetic added per litre.

The oxalic solution is made up of: cold saturated solution 1 part, water 3 parts.

The permanganate solution as used for Fe titrations is too strong. 1·5185 gm. $K_2Mn_2O_8$ per litre affords a solution such that 1 c.c. = 1% Pb."—W. Low. —*School of Mines' Quarterly*, July, 1908, p. 376. (H. A. W.)

Detection of Arsine in Air.—"According to Hebert and Heim, this may be done by taking advantage of the well-known reaction with $HgCl_2$, but when a paper wetted with a solution of this salt is exposed to AsH_3 various other gases which possibly might be present may give a yellow tint which simulates that produced by AsH_3. It has been noted that passing the air through a solution of Cu_2Cl_2 will remove those interfering substances, and that if the air so purified is then passed through a U tube containing a slip of paper moistened with $HgCl_2$ the paper will not show a yellow tint unless AsH_3 be present."—Hebert and Heim.—*Mining and Scientific Press*, June 13, 1908, p. 809. (K. L. G.)

Purification of Mercury.—"Bettel (C.N., XCVII., 158). The author has found that by using a 2% solution of KCy with addition at intervals of a little Na_2O_2 the ordinary impurities may be completely removed from Hg. While the base metals are being dissolved (which in a case cited required about four days) but little Au was removed. After that the removal of Au was fairly rapid, diminishing of course in speed as the proportion of Au became less."—W. Bettel.—*School of Mines' Quarterly*, July, 1908, p. 373. (H. A. W.)

Detection of Mercury in Nitro-Glycerine Compounds.—200 gm. of the explosive were ground up with 400 gm. of French chalk, and 250 c.c. of water added, the whole allowed to stand for 24 hours. The solution was then filtered, slightly acidulated with H_2SO_4, subjected to an electrical current of 1·5 amperes, and the mercury deposited on a gold cathode using a platinum anode.

The electrolysis was conducted for three to six hours. The cathode was detached, washed with water, alcohol, and ether, carefully dried at a low temperature, and weighed, the increase of weight being taken as Hg_2. The cathode was then placed in a test tube and heated to redness, keeping the upper part of the tube cool. The mercury was then volatilised and deposited on the tube. The sublimate which was often invisible to the eye, was made discernible by the addition of a small grain of iodine and gentle warming. The iodine vapour after filling the tube subsides, leaving a red deposit of HgI_2.

As over two hundred tests were to be made, the following more simple and rapid qualitative method was devised:—

200 gr. of the explosive were ground with 400 gr. of French chalk, and placed in a large tube, which had suspended from the cork a piece of silver foil. The tube was heated in at bath at 71°C. for two hours. After cooling, the foil was removed to a test tube, and heated strongly. The consequent sublimate of mercury in the upper part of the tube was detected by the use of iodine vapour as described above. E. A. Mann.—Report of the Department of Mines of Western Australia for the year 1907, p. 225. (A. R.).

Dissolution of Metallic Gold in Hydrochloric Acid in Presence of various Organic Substances.—"Metallic gold dissolves in hydrochloric acid in presence of the following compounds, which are arranged in decreasing order of effectiveness:—Methyl alcohol, amyl alcohol, chloroform, ethyl alcohol, chloral hydrate, phenol, sucrose, glycerol, trioxymethylene, formaldehyde. The dissolution takes place slowly at the ordinary temperature, but much more rapidly on heating, the time of heating having but little effect on the rate of solution. In some cases it was found that the solution of the gold was accelerated by increasing the amounts of acid and organic substance.

The residues obtained by evaporating these solutions on a water-bath give metallic gold on ignition, the loss on ignition differing from that observed with $AuCl_3, 2H_2O$ (loss, 51·6%). Thus, with chloroform, this loss is 45·63; with methyl alcohol, 60·3; with phenol, 65·08; with ethyl alcohol, 78·0; with trioxymethylene, 85·1; and with amyl alcohol, 85·01%.

The results obtained indicate the existence of new compounds and reactions of gold, which are being further studied."—W. D. Auerkieff, *J. Russ. Phys. Chem. Soc.*, 1908, 40, 840-859.—*Journal Chemical Society*, Oct., 1908, p. 859. (W. A. C.)

METALLURGY.

Copper Plate Absorption.*—Edward Halse gives detailed particulars of the gold retained by three sets of silvered plates (apron) in a mill at Sucre, Colombia, which had accumulated during a period of rather more than six years. The ore milled was pyritic and consisted mainly of quartz with some (rarely visible) free gold.

The four plates of the first set were taken up at the end of 26 months and 'burnt.' During that period 4462 tons of high grade ore passed over them, with an average yield of 19·62 dwt. of fine gold, followed by 3453½ tons of tailings, with an average yield of 2·58 dwt. of fine gold per ton (2000 lbs.)

The gold scale from the four plates weighed 937 oz., and the average rate of accumulation was 1·269 dwt. of fine gold per ton milled.

The four plates of the second set were taken up at the end of a similar period, during which 1172 tons of tailings, with an average yield of 1·95 dwt. of fine gold, followed by 2175 tons of low-grade ore, with an average yield of 6·80 dwt. of fine gold per ton passed over them.

The gold scale amounted to 568 oz., which gives an average rate of accumulation of 1·823 dwt. of fine gold per ton milled.

The four plates of the third set were allowed to run for 24 months only. The ore crushed during that period amounted to 2869 tons of medium grade, yielding 11·13 dwt. of fine gold per ton.

The gold scale from these plates weighed 553·68 oz., giving an average rate of accummulation of 2·071 dwt. of fine gold per ton.

The average accumulation of hard amalgam per ton works out at 1·721 dwt. of fine gold. At the Drum Lummon mine, about 12 years ago, Mr. R. T. Bayliss obtained an average of 0·499 dwt. fine gold per ton milled. The grade of ore at Sucre was about double that at Drum Lummon; and, no doubt, other things being equal, the richer the ore the greater the accumulation. From the results at Sucre it would appear that the richest ore gave the lowest rate of accumulation, but this is accounted for by abnormal conditions, interruption and milling of tailings. The author states that it is well known that very low grade ore will gradually remove hard amalgam that has accumulated on plates from rich ores which has previously passed over them, and considers the low rate of accumulation with the first set is thus explained. In the case of the second set of plates there was practically no 'scouring' for the milling of the low-grade tailings was followed and not preceded by the milling of ore (low grade). During the third period the conditions were quite normal. Medium-grade ore had by that time become available, and no tailings were passed through the mill. So that, at Sucre, working under normal conditions, the average rate of accumulation of hard amalgam was as much as 2 dwt. of fine gold per ton, or four times higher than was the case at Drum Lummon. The gold actually absorbed by the plates, and only recoverable by melting them down, was 0·288 grain per ton milled at Drum Lummon, and certainly less than 1 grain per ton at Sucre. The average rate probably does not exceed a fraction of a grain per ton.

---

* See this *Journal*, vol. ix., Aug., 1908, p. 62.

W. E. Thomæ quoted the statement made to him by the manager of a large gold mine on the Rand that 'every new plate in a gold mill will absorb and lock up about 70 oz. of gold;' and remarked that one frequently hears the statement, when a new mill has started with a disappointing return, that it was due to the absorption of gold by the new plates. To what extent is this justified. If the two statements were combined one might jump to the conclusion that the first crushing return of a new mill is reduced by about 70 oz. for each copper plate it contains. Such a conclusion, however, would require very considerable modification.

Consideration of results obtained at the Drum Lummon and Hidden Treasure mines by R. T. Bayliss and A. L. Collins, respectively, appear to justify the following general conclusions:

1. The amount of gold actually absorbed by a copper plate, 4 ft. 6 in. × 8 ft. to 12 ft., is not likely in the long run to amount to more than 8 to 10 oz., and generally not more than half of this, if as much, will affect the first month's crushing returns.

2. The amount of gold retained on a plate in the form of amalgam scale is a variable factor, depending largely on the capacity of the millman and the discretion of the manager.

Though apparently there are cases where the amalgam scale is allowed to accumulate to a large extent, that is not the general practice, and in any case the accumulation during the first month, when the plates are new, will represent only a proportion of the final accumulation.

Thus it would appear that generally the first month's run of a gold mill with new plates is not affected by such a serious loss in locked-up gold as is sometimes attributed to it, or inferred.

Finally, in connection with this subject, the author enquired why Muntz metal plates are not more generally used. They should be cheaper than copper plates, they absorb practically no gold, the amalgam scale does not adhere to them so obstinately, and they are hence more easily cleaned up."—E. Halse and W. E. Thomae.—*Electrochemical and Metallurgical Industry*, Sept., 1908, p. 375. (A. McA. J.)

Operation Costs of the Merrill Process at the Homestake Slime Plant, Deadwood, S. Dak.—"The installation of the Merrill press at the Homestake slime plant at Lead, S. Dak., aroused great interest in the metallurgical world. Not only were mill men interested in the construction and operation of the press itself, which has been previously completely described in *Mines and Minerals* for March, 1907, but they have watched with keen interest the progress made in gradually cutting down its costs of operation. It will be recalled that before the construction of the plant, 1,600 tons of slimes, carrying an average of 80 cents per ton, or a total value of $1,456, were wasted daily.

Mr. Merrill originally estimated that his process would make a 90% extraction of values at an operating cost of 25 cents per ton. This low cost estimate was the subject of much discussion at the time, as it seemed extremely low compared with the costs of other methods. Now, after a year's operation of the presses, in which they have been thoroughly tested, Mr. Merrill has favoured *Mines and Minerals* with the accompanying detailed cost sheet.

Operating Costs per ton, Homestake Slime Plant, March, 1908.—Total Tons Treated, 49,946.

Items.	Labour.	Electric Power & Lighting.	Chemicals.		Other Supplies.	Total.
			Item.	Cost.		
Thickening	00370				·00006	·00376
Transportation					·00030	·00030
Neutralisation	·00625	·00116	Lime, 4·476 lb.	·02236	·00006	·02985
Filling and discharging Merrill sluicing presses	·01040	·00347			·00239	·01626
Dissolving and washing	·02468	·01502	Cyanide, ·31 lb. HCL	·06200 ·02324	·00014 ·01 (for cloths)	·13508
Precipitation, Merrill method	·00281		Zinc, ·127 lb.	·00762	·00026 ·00057	·01126
Heating	·00218				·00537	·00755
Assay Office	·00348				·00518	·00866
Superintendence	·00911				·00043	00954
Miscellaneous	·01272				00117	·01389
Fire protection					·00054	·00054
Refining, bullion expressage, and mint charges					·00864	·00864
Total	·07533	·01965		·11524	·03511	·24533

General Notes.—Hydrochloric acid at $4·30 per carboy, 10 carboys per press; cyanide at 20 cents per lb., lime at ·005 cent per lb.; zinc at 6 cents per lb.;

labour at $3 per 8-hour shift; power at $7·50 per mechanical horse-power per month.

One suit of filter cloths lasts one year. For 24 presses cloth consumption = 2 suits per month = 1 cent per ton of slime treated.

In this table it will be noticed that in the month of March, 1908, selected as a typical month, the plant treated a total of 49,946 tons of slime at a total cost of 24·533 cents per ton. The following total costs per ton are especially noteworthy: Filling and discharging, 1·626 cents; dissolving and washing, 13·508 cents; precipitation, 1·126 cents.

At the Homestake plant, the installation of the fine grinding plant, now nearing completion, has necessitated the immediate addition of two more presses to the original 24, while grading on the hillside is to begin shortly to allow of the enlargement of the containing building for the reception of several more presses."—*Mines and Minerals*, Oct., 1908, p. 141. (A.R.)

THE RAND'S NEW MILLS.—"We hereunder present a table which states the number of stamps and tube mills, weights of stamps, size of tube mills, crushing capacity, and date of commencement of milling of the Rand's new equipments, which have either recently started work, or are about to do so. All the machinery discussed is of new and up-to-date pattern, with the exception of the 60 old stamps of the West Rand Consolidated. It will be noted that the tendency to day is to erect very heavy stamps, and whilst some companies prefer to run their mills by means of steam power, it certainly seems that electricity has come to be regarded as the best method of operating a Rand crushing equipment. It will be observed that a total of 735 stamps and 12 tube mills have either commenced or will soon commence contributing to the Transvaal's gold output."—*South African Mining Journal*, Sept. 19, 1908, p. 802. (G. H. S.)

DIRECT EXTRACTION OF COPPER FROM ITS ORES BY ELECTRICAL METHODS.—"This process, due to T. S. Anderson, has been in operation for some two years, during which time nearly every grade of ore has been treated from 1·74% to 50%. It consists essentially of four operations—namely, crushing, roasting, lixiviation and deposition, no smelting being necessary. After crushing, and being mixed with a given quantity of iron, the ore is roasted in an electric or other type of furnace. The gases are taken to a condensing chamber and are there transformed to sulphuric acid; the roasted ore is then automatically conveyed to the lixiviating tanks, which are so arranged as to facilitate the removal of the residues. The 'coppery' solution then passes direct to the depositing room, where pure electrolytic copper is produced in tubes, sheets, etc., the latter being about 40 in. square and 100 lb. in weight. The copper is ready for the market without any subsequent working and readily obtains the highest price. 600 h.p. is the total power required for all processes for the production of 10 tons of copper per 24 hours. A quantity of the Annadale Company's ore has recently been treated, and pure electrolytic copper produced therefrom at a total cost of £12 10s. per ton."—E. SPARGO.—*Indian Engineering*, Oct. 17, 1908, p. 241. (A. R.)

PITTED GOLD BULLION BARS result usually from adding an excess of nitre and not leaving the "melt" long enough in the furnace to permit the gases generated to escape. When oil or grease is used in the mould to prevent the bar sticking fast, gases evolved from these substances will also pit or roughen the surface of the bar. The best coating is produced by swabbing the heated mould with a lime-wash of the consistence of cream, just before pouring."—*Mining and Scientific Press*, June 13, 1908, p. 799. (K. L. G.)

Name of Company.	No. of Stamps.	Weight of Stamps.	No. of Tube Mills.	Size of Tube Mills.	Crushing Capacity anticipated.		Date of Milling.	Remarks.
					Tons per month.	Duty per stamp per day		
Bantjes Consolidated	35	lb. 1,650	1	22ft. x 5ft. 6in.	10,000	10	Seven months hence.	Electrically driven mill. Co. purpose putting in their own prime movers.
Cinderella Deep	50	1,700	2	16ft. 6in. x 4ft.	12,000	8	End of Sep., 1908	Electrically driven mill.
Geduld Prop.	50	1,650	...	...	...	7	Nov. 1, 1908.	Steam driven.
*Jupiter ...	100	1,700	...	...	...	...	..	...
Knight Central	100	1,700	3	22ft. x 5ft. 6in.	24,000	8	Beginning of 1909 ...	Mill will be steam driven.
Main Reef West	80	...	...	...	...	...	End of 1908 ...	Stamps are being erected at the back of the Con. Main Reef Mill.
Simmer Deep ...	200	1,670	4	22ft. x 5ft. 6in.	...	...	Crushing commenced beginning of Sept.	Electrically driven mill.
Vogelstruis .. Con. Deep	40	1,650	...	..	10,000	...	...	...
West Rand Con.	20	1,700	2	...	16,000	...	Has been crushing for 2 or 3 weeks ...	Steam driven.
	60	1,250	†	...	...	...	...	..

* Joint mill with Simmer Deep. † Old Stamps.

Iron, Sulphur and Phosphorus. — "In the course of an address by Mr. J. E. Stead, of Middlesbrough, on the subject of iron, sulphur, and phosphorus, illustrations were given by means of lantern slides of the various changes which occurred when pig-iron passed from the liquid to the solid state, and the author said that by the Bauman auto-sulphur printing process in less than a minute an approximate amount of sulphur in cast-iron could be arrived at, and that not by an analytical chemist, but by any intelligent boy in a foundry. The method described was the following:—Pieces of ordinary bromide silver paper were soaked in 3% solution of strong sulphuric acid and water. These slips of paper were then laid upon the polished surface, left in contact for one minute, and then removed. Stains of sulphide of silver were thus obtained on the paper corresponding to the actual amount of sulphur in the casting. Another simple microscopic method by which the same lad could make a fair estimate of the amount of phosphorus in pig-iron or castings was given by Mr. Stead, and it was shown that on etching the phosphorus portions with a solution of 20% of nitric acid, those portions being quite independent from the other parts of the metal, they remained bright and clear, all the other parts having a darkened colour. The comparison between Cleveland iron, containing 1·5 % of phosphorus, and hematite, containing ·04 % of phosphorus, was very pronounced. In the former a very large portion of the whole area consisted of white particles which in hematite pig were distributed as minute spots over the surface."—J. E. Stead, British Foundrymen's Association Annual General Meeting, Aug. 4, 1908—*Times Engineering Supplement*, Aug. 5, 1908. (J. A. W.)

Dry Air Blast in Steel Making.—"The introduction of the Gayley dry air blast to blast furnace practice has been previously noted. Experiments have recently been made in America with a view to testing the applicability of desiccated air in blowing Bessemer steel, and some interesting results have been obtained. The trials were conducted at the South Chicago works of the Illinois Steel Company, and the dry air plant supplying the blast furnaces was used. The trials extended over two days, and the average reduction in moisture of the air blown into the converters was from 5·98 to 1·39 gr. of moisture per cubic foot of air. Mr. P. H. Dudley examined and tested the products, and he describes the experiments as of technical and historical importance. The results mark a considerable advance for the Bessemer process and are encouraging in the direction of increased capacity for carrying scrap and other items bearing on cost. A better control of silicon is also obtained, but the most significant feature is found in the improvement in product.

In commenting on these experiments the *Engineering and Mining Journal* states that the results shown may be of great importance in restoring confidence in the Bessemer process, and putting the converters again on a parity with open hearth furnaces as far as quality of product is concerned. The tests were on such a scale, and so carefully made that they may be accepted as fully confirming Mr. Gayley's claims for the advantage of using dry air in the converter as well as in the blast furnace. It is not impossible that in this application may be found the solution of the steel rail question, over which there has been so much controversy during the past two years. If parallel results can be obtained in continuous operation—and there is no apparent reason why they cannot—there will be no necessity to abandon the Bessemer converter and resort to the open hearth for rail steel."—Percy Longmuir.—*Engineering Review*, Sept., 1908. (J. A. W.)

All-sliming Process in New Zealand.—"Mr. Brown has made a still further improvement by attaching to the feed end of the tube mill what he terms an 'injector,' the effect of which is to keep the mill more fully charged, which considerably increases its grinding-capacity and economises power-consumption. Fine-grinding tests on a working scale have lately been made, in order to obtain reliable data in anticipation of the change to the all-sliming process, which we are adopting. These tests have proved most satisfactory, it being found that a tube mill fitted with the patent iron-ribbed liners and other implements will grind from 40 to 45 tons of ore per day to such a fine state that 92 per cent. will pass 200-mesh—that is to say, 40,000 holes to the inch—at a cost for power, consumption of flints, and wear of liners of rather under 6d. per ton. This is an achievement in fine grinding which, as far as I know, has not been approached anywhere else in the mining world. If we had ordinary gold-bearing quartz to deal with, reduction to such a fine state, and subsequent agitation in tall vats, would insure practically a total recovery after a few hours' treatment; but to obtain a high extraction from these silver-sulphide and gold ores, not alone has the ore to be ground to an excessively fine state, but the agitation has to be continued for as many days as hours in the former case. The consequence is that we have to provide an enormous amount of treatment plant in proportion to the tonnage dealt with. Eight tall tanks, each 13 ft. in diameter by 55 ft. in height are all well advanced towards completion."—Waihi Grand Junction—Annual Report 1908.—*New Zealand Mines Record*, Sept. 16, 1908, p. 91. (A. R.)

Chromite for Fire Bricks. — "Chromite or chrome iron, a natural product for some time utilised as a refractory lining for furnaces, must be mixed with a foreign substance, usually fire-clay. According to experiments made at Charlottenburg by Mr. Simonis, an excessive addition of clay may deprive the chromite of its infusibility. A mixture of kaolin and less than 80% chromite is more fusible than pure kaolin. An advantage from the refractoriness of chromite is only obtained when the proportion of kaolin is less than 10 to 15%. The fusion temperature is then considerably greater than that of platinum."—M. Simonis, *Genie Civil*, Oct. 10, 1908.—*London Mining Journal*, Oct. 24, 1908, p. 514. (A. R.)

Use of Iridium and Rhodium Crucibles.—"Sir William Crookes has furnished to the Royal Society (1908) an account of the behaviour during several months' use of pure wrought iridium crucibles. Iridium is as hard as steel, and does not blister or change in weight when heated to redness repeatedly. Heated in a Bunsen gas flame with insufficient supply of air, for several hours, the crucible is uninjured, as the carbon deposited can be entirely removed by burning. Sulphur in the gas is also without harmful effect upon the metal. Prolonged heating with copper renders the crucible brittle whilst hot, but the copper can be burnt off. Fused caustic potash attacks iridium less than platinum, and zinc run in with acid chloride of zinc (soldering liquid) and then heated to boiling only superficially attacks the iridium. Boiling aqua regia, fused microcosmic salt, or other phosphates, with frequent addition of carbon, strongly heated silica or silicates, with a reducing agent, boiling lead at white heat, boiling zinc, and molten nickel, iron or gold, are without effect on the crucible, which, after cleaning, retains its original weight. A rhodium crucible has the advantages of being only half as heavy, of being cheaper, and is practically as resistant as an iridium one."—Sir W. Crookes. —*Indian Engineering*, Oct. 24, 1908, p. 257. (A. R.)

Tungsten and Tantalum. — "Tungsten and metallic tantalum, which some years ago could only be obtained pure with difficulty, even at very great cost, are now produced in considerable quantity, especially by Messrs. Siemens & Halske, of Berlin. Manufacture of these two metals on a commercial scale was only made possible by invention of the electric furnace, in which the metals are melted in a vacuum. Ordinary electric furnaces could not be employed for this purpose, as the carbon of the electrodes, as also the nitrogen and oxygen of the air, would cause combinations at the high melting temperatures of tungsten and tantalum. The melting point of the latter is 2,000° C. to 3,000° C. That of tungsten was only recently fixed definitely by H. de Wartenburg, by aid of the new type of electric furnace and the Wanner optical pyrometer. According to his researches, tungsten melts at 2,800° C. to 2,850° C., and has thus probably a higher melting point than all other substances except carbon. Consequently, the data supplied by Mr. Riche (3,000° C.), and Messrs. Wardner and Burgess (3,080° C.), hitherto taken as basis, are to be rejected. The most important application of these two metals will be for incandescent lamp filaments, but tantalum in particular becomes more and more interesting, owing to its hardness and tenacity, which are not surpassed by any other metal."—*Revue d'Electrochimie et d' Électrométallurgie*, July.—*London Mining Journal*, Sept. 19, 1908, p. 361. (A. R.)

MINING.

Elastic Iron Pit Props.—"A new pattern of Nellen's elastic iron props for pit use is described by Hecker in *Glückauf* for Oct. 24, as having been used, satisfactorily, at the Heinrich shaft, Neu-Essen colliery. The prop consists of a steel tube, divided longitudinally and held together by bolts passed through flanged bands, one of the halves being divided into two lengths, with a space of about an inch between them, this space being covered by a band when the prop is in position. In the top end of the tube is inserted a timber prop, the head of which is hollowed out to fit the capping timber and surrounded by a steel band to prevent splitting. Below this timber the tube is packed with sand for a certain distance, and underneath this, again, are placed a number of peat discs, filling up the tube, the bottom of which is closed by a wooden plug. The sand and peat form an elastic backing to the timber prop which is thus able to give a little under the roof pressure, without bending. To recover the props when the worked-out portion of the mine is filled with goaf packing, the band covering the slit is removed, and a little of the sand is taken out with the pick, whereupon the timber prop sinks enough to free it from the capping timber, and the prop can be withdrawn. Another type of telescopic prop—the Hinselmann—used at the Rheinpreussen colliery, consists of two tubes, fitting one in the other and pierced with slots, whose relative position is such that the prop can be lengthened to any desired extent (within the limit of its total length) by driving iron wedges into these slot-holes in succession. All that is necessary to recover the props is to knock out the wedges."—*London Mining Journal*, Oct. 31, 1908, p. 548. (A. R.)

Coal Dust Ignition.— "At a recent meeting of the Midland Institute of Mining, Civil and Mechanical Engineers, held at Doncaster, England, Mr. J. Neal read a paper describing the ignition of coal dust in the Beeston seam at the Middleton Colliery. The seam was a very dry one, and dust accumulated very rapidly. Early in September, 1907, a deputy had opened a safety-lamp at the lamp-station. A train of full tubs, drawn by a pony, was passing, and raised a considerable quantity of dust. Just before the train reached the lamp-station the deputy removed part of the burning "snuff" from the wick of the lamp, and it fell harmlessly to the floor. As soon as the train had passed he knocked away the remaining portion of the snuff, and as this fell to the floor there was an ignition of the coal dust. The flame rose to a height of about $2\frac{3}{4}$ ft., and, spreading over a width of 3 ft., followed the train with a peculiar rolling motion, apparently corresponding with the successive clouds of dust raised, making a peculiar hissing sound. The train stopped about 45 ft. beyond the lamp-station, and when the train reached the last tub it ascended to the roof, returned along the upper portion of the roadway, and finally extinguished itself within 3 ft. or so of the point of ignition. The colour of the flame was described as being similar to that of a candle or oil-lamp. Mr. Neal observed that mining engineers had thought that nothing short of an ignition or an explosion of gas, or gas and dust, could cause a coaldust explosion, and that was his opinion until this incident occurred." —J. Neal.—*New Zealand Mines Record*, Sept. 16, 1908, p. 53. (A. R.)

MISCELLANEOUS.

Storage of Petrol.—"A new plan for storing gasolene or petrol depends upon the fire-stopping property of wire gauze that gives safety to the Davy mining lamp. A gauze tube is inserted in the opening of the can or tank, extending to the bottom, and the orifice is then sealed up by a plug held in place by fusible solder. If fire occurs near or around the tank, the solder melts and the plug is blown out, when the vapour escaping through the gauze tube burns quietly without explosion. In a test of the method, twelve out of seventeen ordinary cans filled with volatile oil exploded on contact with fires throwing the burning liquid in every direction, but twelve cans fitted with the safety device failed to explode and the liquid burned quietly and harmlessly."—*The Canadian Mining Journal*, Aug. 1, 1908, p. 360 (C. B. K.)

Steel Bands for Leather Belts. — "Steel bands or belts, as a substitute for ordinary leather belts or rope drives, have been introduced by a factory of Charlottenburg, Germany. Such belts may be reduced to about one-sixth of the size required for leather belts, they do not stretch, pulleys may be made narrower, and in some cases may be smaller. Either ordinary pulleys or pulleys with a special covering to increase friction may be used. A belt 4 in. wide and $\frac{1}{5}$ in. thick transmitted 200 to 250 h.p. at a belt-speed of 5,400 ft. per minute, and tests have shown that steel belts may run 12,000 ft. per minute." —*The Canadian Mining Journal*, Sept. 15, 1908, p. 467. (C. B. K.)

Electric Lighting in Glasgow.—"The Electricity Department of the Glasgow Corporation have just published their accounts for the financial year 1907-8. The total capital expenditure up to the end of the year was £1,685,084, or £62·4 per kilowatt capacity of plant installed. When it is remembered that a large turbo-generating set can be purchased for less than £5 per kilowatt of capacity it will be understood that the engine and generator are not the only items in an electrical scheme. The principal items of capital expenditure include land and buildings, which account for £227,698, or 13·47% of the total; mains and cables account for 53·48%; and machinery and plant for 28·78%. These figures do not include the generating stations for tramway power, which are kept distinct from the light and power stations. The average receipts per unit were 1·854d., and were expended as follows:—0·49d. for generating costs, 0·109 for distribution costs, and 0·245 for management (including rent, rates, and taxes), and 1·003d. for capital charges, leaving a balance of 0·007d. It will be noted that the capital charges amount to more than half the gross cost of production, and are three times as large as the fuel costs."—*Journal of Royal Society of Arts*, Oct. 30, 1908, p. 1038. (C. B. K.)

Arseniated Hydrogen Poisoning.—"Poisoning by arseniated hydrogen is relatively rare, although this gas is often produced in chemical laboratories and various industries like that for extraction of gold by the cyanide process. However, amongst 60 specified cases of poisoning, only 14 related to workers in metals; 17 were ascribed to scientific researches; 7 manufacture of aniline dyes; while the production of hydrogen from zinc scrap and impure sulphuric acid accounted for 10 cases amongst Italian balloon dealers. The first known accident occurred at Munich in 1815. Gehlinehmnit, having prepared this gas, endeavoured to recognise it by its odour, and inhaled sufficient to cause his death nine days later. Arseniated hydrogen poisoning differs in many points from that due to other arsenical compounds. According to a study by Mr. Gardner, in *La Clinique*, the chief symptoms are pains in the region of the loins, jaundice, and hæmoglobinuria. The effects of inhalation are not felt until some hours after inhalation, according to the amount inhaled, and their appearance is sudden—sensation of cold, shivering, rapid prostation, vertigo, etc. The purely symptomatic treatment is abundance of fresh air and oxygen, provocation of diuresis, and maintenance of the patients' strength. In 60 cases of poisoning there were 20 deaths, after an average of eight days. Cure is complete if effected."—*Revue Scientifique*, Oct. 27, 1908.—*London Mining Journal*, Nov. 7, 1908, p. 577. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

The Metallurgy of Common Metals. By L. S. Austin. (San Francisco: *The Mining and Scientific Press.* p. 407.)

"This book is of a somewhat general character, and the metals included are gold, silver, iron, copper, lead and zinc. Before passing on to a consideration of these metals, the author gives an excellent outline of metallurgical preliminaries, and a really good chapter on roasting. The metals enumerated above are then dealt with in separate sections, and the three on gold, silver and copper certainly show the author at his best. The section on zinc is remarkably short, and is, in fact, simply sufficient to whet the appetite for more. The section on iron is limited to the production of pig-iron, and it is perhaps a pity that the author has not included an outline of the further treatment of pig-iron. The concluding sections represent a special study of refining and commercial conditions. The latter, though intended for American readers, is not without value to a British audience. In fine, the book does not pretend to exhaustive detail, but rather gives a salient description of the metallurgy of the more familiar metals." — *Engineering Review*, Sept. 1908. (J. A. W.)

Modern Practice in Mining. Coal, Its Occurrence, Value and Methods of Boring. By R. A. S. Redmayne, M.Sc., M.Inst.C.E., M.Inst. M.E. (London: Longmans, Green & Co., 39 Paternoster Row. p. 196 + vi. Price 6s. net).

Compressed Air Plant for Mines; The Production, Transmission and use of Compressed Air, with Special Reference to Mine Service. By Robert Peele, Mining Engineer and Professor of Mining in the School of Mines, Columbia University. (New York; Wiley & Sons.)

"While the literature bearing upon this branch of compressed air service is by no means small, it is for the most part scattered through technical periodicals and the transactions of engineering societies, and therefore not readily accessible to those who are out of convenient reach of engineering libraries; that is to say, the majority of those who need and would appreciate the garnered experience of practical men. In the present instance, the student has the advantage, which is a very considerable one, resulting from the fact that the material has been collected, collated, sifted, and finally adapted to the special requirements of mining engineers, by one who is an acknowledged leader in the profession. The first portion of the book, under the heading, 'Production of Compressed Air,' deals with current practice in the construction and operation of compressors, the types of compressor suitable for different kinds of service, heat losses occurring in air compression, and the various forms of valves, valve-motions, and governing and unloading mechanisms, that constitute prominent features of modern compressor practice. In the second part the transmission and use of compressed air are dealt with, as applied to various departments of mining work, including machine drills, pumps operated by compressed air, and mine haulage by compressed-air locomotives. The book is excellently printed, and the 112 illustrations by which the letterpress is elucidated are well brought out by the heavy calendered paper employed. There is also a very serviceable alphabetical general index, which cannot fail to be appreciated."—*Australian Mining Standard*, Oct. 21, 1908, p. 454. (A. R.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 472/08. Edward Lloyd Wynyard Bellhouse (1), Reuben Jones (2). Improvements in or relating to pneumatic and steam-driven rock drills. 27.11.08.

(P.) 473/08. Henry Johnston. An improved chair for railway and like rails. 28.11.08.

(P.) 474/08. John Davis (1), Peter Miller (2). Improvements in drill heating furnaces. 28.11.08.

(P.) 475/08. Alexander John Arbuckle (1), Alfred Osborne (2). Improvements in means for separating crushed ore products or other comminuted solid matter from liquid. 1.12.08.

(C.) 476/08. Robert William Felton. Improvements in or relating to wire-rope jockeys. 1.12.08.

(P.) 477/08. Seymour Merrington. Improvements in packing for glands and the like. 2.12.08.

(P.) 478/08. Frederick Ruther. Spring lever stamp battery. 2.12.08.

(P.) 479/08. George Robert Simms. An ore conveyer. 4.12.08.

(C.) 480/08. William L. Imlay. Process of extracting precious metals from their ores. 4.12.08.

(P.) 482/08. J. M. Holman (1), J. L. Holman (2). Improvements in rock drilling machines. 8.12.08.

(P.) 484/08. Hans Charles Behr. Improvements in stamp actuating means. 8.12.08.

(P.) 486/08. Bernard G. C. Ockerse. An improved oil extractor. 9.12.08.

(P.) 487/08. Alexander John Arbuckle. Improvements in tanks or vessels for separating comminuted solid matter such as pulverised ore from liquid. 11.12-08.

(C.) 488/08. Jene Lassen le Cour. Improved means for regulating polyphase commutator machines. 11.12.08.

(P.) 489/08. Henry Thomas Durant (1), Henry Livingstone Sulman (2) Hugh Fitsalis Kirkpatrick Picard (3). Improvements in the treatment of zinc box precipitate. 11.12.08.

(P.) 491/08. James Chalmers Johnstone. Guide clock for battery stamps. 12.12.08.

(P.) 492/08. Sydney Arthur Cole. Improvements in rock drilling machines and like reciprocating engines. 12.12.08.

(C.) 493/08. Edward Brice Killon. Improvements in or relating to wheels. 15.12.08.

(P.) 494/08. H. A. Stockman. Improvements in apparatus for recovering precious and other metals, and precious stones from matter containing the same. 17.12.08.

(P.) 495/08. T. H. Mutch. An improved plant for use in cyanide or like processes of extracting metals from their ores. 18.12.08.

(P.) 496/08. H. M. Carter (1), L. B. de Laitte (2). Improvements in the means for preventing accidental movement of rotating parts. 18.12.08.

(P.) 498/08. J. H. Fortescue. Roller mill concentrate crusher. 19.12.08.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

Adeney, G. B., *l/o* Johannesburg; Siamese Tin Syndicate, c/o Messrs. Adamson, Gilfillan & Co., Penang, Straits Settlement.
Anderson, J. S., India; c/o A. J. Barlow, Samox, Victoria Road, Eccles, Manchester, England.
Bawden, E. R., *l/o* Nyassaland; Poldice House, St. Day, Scorrier, Cornwall.
Bennett, R. W., *l/o* Knights; c/o W. C. Geach, Consolidated Main Reef G. M. Co., Ltd., Maraisburg.
Blume, E. E. A., *l/o* Namaqualand; P. O. Box 908, Johannesburg.
Brickhill, H. G., *l/o* Germiston; Nigel Deep, Ltd., P. O. Box 50, Nigel.
Burnett, Digby V., *l/o* London; Pigg's Peak Development Co., Ltd., Swazieland, via Barberton.
Bosanquet, F. G., *l/o* Knights; c/o Rev. G. Bosanquet, Lyde Vicarage, Hereford, England.
Cannon, A. B., *l/o* Eureka City; Barberton.
Clennell, J. E., *l/o* London; Société de Mines de la Belliere par Montrevault, Maine et Loire, France.
Curtis, A. H., *l/o* Bedford; 822, Salisbury House, London Wall, London, E.C.
Deakin, C. E., *l/o* Johannesburg; Messina Development Co., Ltd., Messina.
Gerlings, H., *to* 60, Derby Road, Johannesburg.
Gibb, J. A. P., *l/o* Benoni; P. O. Box 2927, Johannesburg.
Graham, W. H., *l/o* Johannesburg; Jumbo Mine, Mazoe, via Salisbury, Rhodesia.
Hamilton, W., *l/o* W. Australia; 45, Scarsdale Villas, Kensington, London, W.
Holmsen, S., *l/o* Luipaardsvlei; c/o Messrs. Frolich et Cie, Christiania, Norway.
Hunter, Chas., *l/o* Rhodesia; Lisbon Berlyn G. M. Co., Pilgrims' Rest.
Lawrie, Everett, *l/o* India; Siamese Tin Syndicate, c/o Messrs. Adamson, Gilfillan & Co., Penang, Straits Settlement.
Miller, J., *l/o* Ringarooma; Mangana, Tasmania.
Morrisby, A. Clayton, *l/o* Gatooma; Hamburg Mine, Golden Valley, Rhodesia.
Olds, H. F., *l/o* Mafeking; Chapel Road, St. Just, R.S.O., Cornwall.
Ransom, W. R., *l/o* Nigel; Transvaal Hotel, Krugersdorp.
Reynolds, J. P., *l/o* Cleveland; Milky Way Gold Mine, Golden Valley, Gatooma, Rhodesia.
Robinson, R. W., *Journals to* P. O. Box 5217, Johannesburg.
Rouillard, R. A., *l/o* Johannesburg; Klipval Mine, P. O. Louwsburg, via Vryheid, Natal.
Saner, C. B., *to* P. O. Box 1145, Johannesburg.
Scaer, V. E., *l/o* Lomagundi; Antelope Mine, Matopos, Rhodesia.
Sharp, E. Miles, *l/o* Johannesburg; Warren Hill Syndicate, P. O. Ariston, Klerksdorp.
Smythe, J. J. R., *l/o* Johannesburg; Warren Hill Syndicate, Ariston P. O., via Klerksdorp.
Strout, E. A., *l/o* Knights; Guanajuato Development Co., Apartado 25, Guanajuato, Mexico.
Ward, J. P., *to* Meyer and Charlton G. M. Co., Ltd., P. O. Box 1127, Johannesburg.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. IX. JANUARY, 1909. No. 7.

Proceedings
AT
Ordinary General Meeting, January 16, 1909.

The Ordinary General Meeting of the Society was held in the Chamber of Mines, on Saturday evening, January 16th, Mr. R. G. Bevington (President), in the chair. There were also present:—

49 Members: Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, K. L. Graham, A. Richardson, C. B. Saner, G. O. Smart, Prof. G. H. Stanley, H. A. White, Prof. J. A. Wilkinson, S. H. Pearce, W. A. Caldecott, F. F. Alexander, S.Beaton, W. Beaver, W. K. Betty, E. A. Blume, W. Carr, M. H. Coombe, J. P. Creed, E. A. Furner, A. D. Gilmour, G. Goodwin, A. Heymann, C. B. Hilliard, Chas. Hunter, A. J. Johnston, T. Johnson, T. S. Jolly, T. W. Jordan, J. Kennedy, T. Kerr, G. A. Lawson, J. Lea, G. Melvill, P. T. Morrisby, W. D. Morton, W. C. Mossop, S. Newton, W. Nicklen, F. B. Ogle, N. Rodgers, O. D. Ross, R. S. G. Stokes, A. Thomas, J. E. Thomas, O. Tonnesen, J. F. Walker.

11 Associates: Messrs. A. R. Adams, C. F. Bayley, J. Bowyer, C. L. Dewar, A. N. Locke, C. B. Simpson, W. A. C. Tayler, G. G. Thomas, C. Toombs, W. Waters, and J. Whitehouse.

10 Visitors and Fred. Rowland, Secretary.

The minutes of the previous ordinary monthly and special general meetings, as printed in the December *Journal*, were confirmed.

NEW MEMBERS.

Messrs. F. B. Ogle and O. D. Ross were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that the candidates for membership had been duly elected, as follows:—

FERGUSON, JAMES FREDERICK, P. O. Box 2309, Johannesburg. Metallurgist.

STREETER, JOHN RADFORD, Village Main Reef G. M. Co., P. O. Box 1091, Johannesburg. Mine Captain.

The Secretary: Since the last meeting of the Society the following associates have been admitted:—

ADAMS, ANGUS ROBERT, Lancaster West G. M. Co., Ltd., P. O. Box 360, Krugersdorp. Assistant Assayer.

BERRY, WILLIAM JAMES, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Carpenter.

DUNCAN, WILLIAM SCOTT, East Rand Proprietary Mines, Ltd., P. O. Box 66, East Rand. Fitter.

JONES, WILLIAM THOMAS, P. O. Box 28, Pilgrims' Rest. Mine Overseer.

NEAL, WALTER, Las Dos Estrellas, Apartado 34, El Oro, Estado de Mexico, Mexico. Superintendent Cyanide Department.

NIXON, GEORGE HENRY, P. O. Box 11, Pilgrims Rest. Mine Overseer.

READ, FREDERICK GEORGE, Transvaal Gold Mining Estates, Ltd., Pilgrims' Rest. Mill Manager.

RICHARDSON, JOHN STEPHEN, 25, Hunter Street, Yeoville, Johannesburg. Mining Engineer.

SCHOLTZ, EWALD VOS, P. O. Box 28, Pilgrims' Rest. Mining Student.

SYMONDS, MELVILLE CARTER, Driefontein Cyanide Works, P. O. Box 66, East Rand. Cyanider.

THOMAS, GEORGE GORDON, Wolhuter G. M., Ltd., P. O. Box 1160, Johannesburg. Chemical Engineer.

GENERAL BUSINESS.

The Secretary: I have to explain, with much regret, that the *Journal* of the Society, for the first time in ten years, has not been published prior to the meeting. This is due to three consignments of paper having come out which are not in accordance with the sample. The proper paper has now arrived in Durban, and is leaving by the fast train to-night, owing to the kind assistance of the Natal Railway agent here. I hope to send out the remainder of the November *Journals* about Tuesday, and the December issue as soon as possible afterwards.

The President: On behalf of the members of our Society I beg to tender our thanks to the Agent of the Natal Government Railway for his kind services in thus helping us in the interests of Science. I regret to have to tell you that our Honorary Treasurer, Mr. Littlejohn, has been very seriously ill. He has lately undergone an operation for appendicitis, and the latest news is that although not out of danger, he is getting on well. Our reporter, Mr. Greenwood, is also unfortunately too ill to be present. He has had a very bad attack of pleuro-pneumonia, but I am glad to say that he also is getting on well.

I should like to state that this will probably be the last meeting we shall hold in this Council Chamber. You will have seen a note in the Agenda paper as to the removal of the offices of the Society. We have been able to obtain, through the kindness of the Council of the Transvaal University College, a very good office on the ground floor of the new building, at what may be termed a very nominal rent. That, of course, will mean a great saving to our Society, and our meetings will in future be held in the Lecture Theatre, which will give greater opportunity to those members who wish to read a paper on a subject for which it is advisable and perhaps necessary to have a practical demonstration. The table will be supplied with electricity, gas and water. It will, therefore, only be necessary for members to bring along their apparatus, in fact, I daresay, even this item might, through the kindness of some of the professors, be arranged. The Transvaal University College has also apportioned one room as a Common Room, to be used by members of the various technical societies. This room will be supplied with various technical journals, and any members who are in town during the hours when the Institution is open will be able to drop in there to rest and read, and meet one another ; and thus, we may hope, become better acquainted. You will also note that the "Seymour" Library has been removed to that building, so that members wishing to consult volumes in that library will find them there. It is very satisfactory to find all these matters centred in the one Institution, and I think we must be glad that the powers that be have seen fit to establish such a splendid building.

THE SILVER COATING OF AMALGAMATING PLATES.

Mr. F. T. Mumford (*Member*): I note that Mr. Caldecott recommends pouring mercury into a solution of silver nitrate containing free nitric acid with a few pieces of bright iron present, in order to form silver amalgam. He remarks, "in the process an iron mercury couple is formed, which serves to deposit the silver present in solution upon the surface of the mercury which immediately absorbs it, whilst a corresponding amount of iron is dissolved." As the same action takes place without any iron being present, I feel inclined to doubt this explanation. If some double crystallised silver nitrate be dissolved in water and a little mercury added, the mercury precipitates the silver, forming silver amalgam if there be an excess of mercury present. I have always been under the impression that the action was represented by the equation $AgNO_3 + Hg = HgNO_3 + Ag$, and that the action was a simple one of displacement, the iron having probably no more effect than a glass rod. I have made large quantities of silver amalgam by simply dissolving bar silver in nitric acid and adding an excess of mercury, and have found no difficulty. Afterwards I have evaporated the liquor to dryness and retorted, recovering the mercury dissolved. The whole operation can be done in a day. I should like to point out that £1 worth of silver coin contains only about 3 oz. of silver ; coin is, therefore, the last thing that should be used, as silver nitrate can be obtained locally for chemical purposes at a cheaper rate, saving the cost of the nitric acid, and I have experienced no difficulty in getting bar silver at about 2s. 6d. per oz. A clean and cheap method of making silver amalgam is the electrical, using a cyanide solution as electrolyte, a bath of mercury as cathode, with the silver bar as anode.

The President : In the unavoidable absence of Mr. Clark, Prof. Wilkinson has kindly undertaken to read the following paper.

PRECIPITATION FROM CYANIDE SOLUTIONS BY ZINC SHAVINGS AND DUST : A COMPARISON OF RESULTS AND COSTS.

By Allen J. Clark (Associate).

The relative advantages of the various methods employed for the precipitation of gold and silver from cyanide solutions have been frequently discussed, and the details of the operations described ; but it not unnaturally happens that few opportunities offer of comparing two methods of precipitation in the same plant, and on a working scale, with solution practically identical in composition.

It has been my privilege to make such a comparison of the "zinc shaving" and the "zinc dust" methods, and the following summary of the results obtained, is submitted with the hope that it may be of value.

While it is not my aim to describe in detail the equipment and manipulation ; some explanatory notes, chiefly as to the nature and derivation of the solutions, are desirable, and are presented. The No. 1 sand plant of the Homestake Mining Company, the oldest of the three secondary plants now in operation, treats the "sandy" portion of the tailings delivered from 640 stamps, from 1,700 to 1,800 tons a day, this sand being leached in vats 44 ft. in diameter and 9 ft. deep, holding about 600 tons of sand. Since the beginning of operations in 1901, zinc dust has been used for the precipitation of the major portion of the solutions, and was the only provision made for precipitation at first, it being thought that sufficient barren solution could be wasted daily, to maintain the stock solution at a normal value. It soon appeared, however, that cyanide would be saved by separating the "first" and "final" solution from that derived from the intermediate stages of treatment. Thus, when the first effluent solution displacing the retained water carries but a trace of cyanide, a gold value of 5 cents per ton appears, and this value increases rapidly until when a strength of ·03% to ·04% is reached, gold to the amount of $1.20 to $1.60 per ton is carried in solution. Similarly, in the final washing, when the strength of ·05% is reached, the assay value is about 15 cents to 20 cents per ton, which decreases until at strength ·02% the value is 4 cents to 7 cents per ton. These solutions were kept separate from the solutions of greater strength and value, and precipitated in zinc boxes which were provided for the purpose, the barren solution from these boxes being run to waste. The average strength of the mixture of this "first" and "final" solution is about ·025%, while the barren solution previously wasted ran from ·06% to ·07%. It was found that, to insure the precipitation of this solution by zinc shavings to a barren assay value of 2 cents per ton, the strength of the pregnant solution had to be maintained at ·03%, and this was accomplished, first by turning in the "final" solution at a strength of ·06% or more, but later by adding barren solution from the main precipitation system, of an average strength of ·07%, and an assay value of 1 cent or 2 cents per ton.

In 1906, experiments in precipitating this solution with zinc dust were undertaken, and on their successful conclusion the necessary equipment was provided, and the installation was ready to operate early in 1907, when a serious fire in the mine caused a suspension of operations, and later led to a campaign of several months' duration upon ore derived from the surface and upper levels, with the result that the solutions departed widely from their usual composition. On this account, the year 1907 was an abnormal one, and the results obtained during this time are omitted from the comparison in which the results by zinc dust during the first nine months of 1908 are compared with the zinc shaving results during 1906, which are representative of the best results obtained by the latter method.

The equipment provided for zinc dust precipitation consists of two tanks, each of 70 tons capacity, and a Merrill precipitation press of 16 frames, which is placed about 15 ft. lower than the tanks, the solutions flowing from the tanks through the press by gravity. By the use of two tanks, one of which is filling while the other is discharging, a more uniform solution is obtained, and a thoroughly accurate sampling and measuring of the pregnant solution is made possible, but this provision is in no way essential to the successful operation of the process. Indeed, in a recent installation of this method at our slime plant, provided for the precipitation of similar "waste solution," it was not practicable to provide two tanks, and here a single receiving tank is used, and a continuous flow to the press is maintained for a period of 30 days, the interval between "clean-ups." The zinc dust is introduced in the form of an emulsion to the pipe running from the tanks to the press, and the precipitation would seem to be almost instantaneous, as the length of this pipe is not more than 20 ft.

For purposes of comparison, all supplies are figured on the prices of 1907. Labour is paid at the rate of $2.50 and $3 for a shift of 8 hours. The ratio of gold to silver in the solutions under consideration is about 2 : 1 by weight; the protective alkalinity is from ·3 lb. to ·5 lb. of CaO per ton. All costs are included up to the delivery of the precipitate at the refinery.

Monthly Averages of Results and Costs.—

Year.	1906.	1908.
Months averaged	12	9
How precipitated	Shavings	Dust
Cubic contents, boxes	142 cub. ft.	—
Filtering surface, press	—	230 sq. ft.
Maximum flow per hour, tons	12	14
Minimum strength of solution precipitated	0·03%	0·015%
Average strength	0·03%	0·0242%
If below minimum strength, strengthened by addition of	Barren works solution, of strength ·07%	The lowest strength encountered, ·015% is successfully precipitated
Tons precipitated	5,909	5,585
Dollars recovered, total	$1,513·	$1,513·
Dollars recovered, per ton solution	0·2560	0·3060
Average barren assay	0·0214	0·0178
Per cent. precipitated	91·7%	94·4%
Pounds of precipitant used, pounds		
Total	1,019	718
Per ton solution	0·1724	0·1285
Per dollar recovered	0·6735	0·4192
Value of precipitate per pound	$2·35	$4·03

Cost of precipitant	129·75	51·15
Cost of labour	97·50	19·75
Cost of miscellaneous supplies	16·03	6·03
All costs, total	$243·28	$76·93
All costs, per ton solution	·0412	·0138
All costs, per dollar recovered	·1608	·0449
Pounds cyanide, in solution, wasted after precipitation	3,545·	2,703·
Value of same	$709·00	$540·60

Summary.

	Shavings.	Dust.	Saving by Dust.
All costs, labour and supplies	$243·28	$76·93	$166·35
Gold discharged in barren solution	126·45	99·41	27·04
Value of cyanide in wasted barren solution	709·00	540·60	168·40
Totals	1,078·73	716·94	361·79

I am aware that these solutions are unusually low both in strength of cyanide and assay value, but believe that could a comparison be made of the two methods in the precipitation of the main working solutions of the plant, the results would be in favour of the zinc dust method.

The President: I am sure we are very glad to have had this paper placed before us. The precipitation of gold from cyanide solutions by the alternative methods of using zinc in the form of dust or shavings seems to be engaging a good deal of attention, and it is naturally of much importance to the industry that the points in favour of or against one method as compared with the other should be thoroughly discussed, and I would like to propose a hearty vote of thanks to the author for his paper.

This was agreed to.

The following paper was then read :—

WHITE LABOUR IN MINING.

By Tom Johnson (Member).

In my paper, "Notes on Mining," recently read before this Society, in the remarks on labour, I expressed the opinion that more white labourers could be used in the mines, also that on shovelling a few could be used without increasing costs. There are some who say that whites cannot shovel as much rock as natives. If that were so I should wish to change my colour, but fortunately for me I have seen it proved more than once that whites can, if they choose, do much better at shovelling than natives. If our system of handling rock were more adapted to the use of white labour and we could get hold of a few men who had handled white labourers shovelling rock in other countries, we might have a chance of seeing what could be done.

The Chinese, owing to the contract system, did better work than the natives, but they did not get anywhere near what I have seen whites do. It has been my experience that the main troubles with white unskilled labour for shovelling have been the spasmodic nature of its employment; the men's dislike of the work, not alone because it is hard, but because it is the fashion to look at it as only Kafir work, and they are looked down on by the aristocratic skilled man. Another thing is that, of the class of men who have tackled shovelling, a great proportion have never done work like it before, and it breaks their hearts to see other men getting their living in a much easier manner. It is a fact that many of the so-called skilled miners do look down on the poor unskilled man, forgetting how short a time has elapsed since they themselves were the same. Bosses also have not looked at the unskilled men with too friendly an eye, due to the trouble of handling them. I do not think these men should be looked down upon, as there is no loss of dignity in a man doing what work comes to his hand so long as it is honest. Some may think I am a bit of a crank in mentioning white unskilled labour, but, for my own part, outside of any wish to see whites working and earning something, instead of starving, I think we are too much in the hands of the natives. Just imagine where we should be if they took it into their heads to strike. I guess we should be much worse off than in the last strike; some may pooh-pooh the idea of a strike amongst the natives, but stranger things have happened.

In the case of getting more white labour on machines, I am going to try and prove that the number of whites on machine stoping can be increased, with increased benefit to men and employers. I hope members will follow the figures closely, and see whether my assumptions are correct or not. It is some time since I had an opportunity of seeing or getting out figures that are necessary for the proposed example, but I do not think I shall be far out, and if I am, I am sure members will let me know.

For example, we will assume that 1,000 tons of rock are wanted per shift from a stoping width of 4·5 ft., say 74 fathoms. The average fathomage is, say, ·5 fathom per machine per shift. I know that some men will break, say, ·8 of a fathom per machine shift, and some as low as ·3. We know that many men drill footage enough, and that if it were done more intelligently they would break much more rock; as a matter of fact, the difference in footage drilled between the good and poor miner is, on an average, small. Now, what I want to do is to try and put these men of different ability into different positions. I would take the man who is breaking the ·8 of a fathom and put him over 10 or 12 of these men who can *drill*, but not *break* ground (one of these men to be on each machine), and, say, 10 or 12 machines. I am not now worrying that they should all go on one face; two faces can be used so long as they are reasonably close.

Now let us see what we could expect from the re-arrangement, after the men had got used to the system. In the present system we need for breaking 74 fathoms, 148 machines at ·5 fathom per machine with, say, 3 machines per man, requiring, say, 50 men, and at 7 natives per man, 350 natives. If the fathomage price were 45s. per fathom, the cost would be £166 10s.

The costs would work out approximately as follows:—

I take gelignite at 20 lb. per fathom, although it would probably average a little higher. Natives at full cost.

Natives, 350 at 3s.	£52	10	0
Gelignite, 30 cases at 40s. 6d. ...	60	15	0
Fuse, 120 coils at 4½d.... ...	2	5	0
Detonators, 6 boxes at 4s. ...	1	4	0
Light, 148 at 6d.	3	14	0
	120	8	0
	166	10	0
	46	2	0

£46 2s. between 50 men would come out at, say, 18s. 6d. per shift.

In the proposed system, we take the men who can break ·8 fathom per machine, one man to look after, say, 10 machines with 10 white drillers. I should expect him to bring up the average to ·7 fathom with the same drilling as is done now, but if we used a rigging and blasting gang I should say that the drillers coming on with machines already rigged up would do 50% more drilling, which would raise the fathomage per machine to 1·05, say, a fathom per machine. Then for 74 fathoms we require 74 machines, 74 white drillers, and, with 1⅓ natives per machine, 99 natives and 7½ white bosses or supervisors. Then we want half of 7 blasters with, say, 30 boys each, which is 3½ whites and 105 natives. This is allowing 3 boys to take down and clear and then re-rig one machine for next shift, time 3 hours. Total number is 85 whites with 204 natives. The fathomage and cost is the same, 74 fathoms at 45s., £166 10s.

The costs would work out approximately as follows:—

Gelignite would be, with the better judgment in placing holes and charging, 16 lb. per fathom, say.

Natives, 204 at 3s.	£30	12	0
Gelignite, 24 cases at 40s. 6d. ...	48	12	0
Fuse, 90 at 4½d.	1	13	9
Detonators, 4½ at 4s.		18	0
Light, 90 at 6d.	2	5	0
	84	0	9
	166	10	0
	81	9	3

£81 9s. 3d. between 85 men would come to 19s. 2d. each per shift, or 8d. per shift better than the other system.

So that by re-arranging the labour force on machines there would be a gain to everyone; the employers would have less capital charge for machines and compressors, less maintenance and a number of natives liberated for other work; the men would have a better average wage and a more regular one, and there would be more of them at work in any mine, besides those who would be necessary to look after the natives who would be liberated for other work. I would like to point out that there is also a gain to us all and the country, in getting more people to work, for the more there are helping to pay revenue, the less per head will the taxes amount to (keeping down the unit expense again); it would also make room for some of the poor fellows now tramping the veld, who have to be kept by somebody.

If members cannot agree with me in the assumption that it would be as cheap to run one white per machine in the manner described as it is to run one white on three machines, I may be better able to carry them with me in the following example. For instance, say, a white man and seven natives are running three machines stoping, each machine breaking 7 tons, equals 21 tons per shift. The costs would be somewhat as follows:—

White wages... ...	£1	0	0	
Coloured wages ...	1	1	0	
Gelignite, 30 lb. ...	1	4	4	
Fuse, detonators, lights, etc.		3	0	
Air maintenance, drills, etc.	2	2	0	= £5 10 4

which would equal a cost of 5s. 3d. per ton. If a white man and five natives were put to run two machines I should expect to get at least 1 ton per machine more, making 16 tons per shift.

The costs in this case would be as follows:—

White wages... ...	£1	0	0	
Coloured wages ...		15	0	
Gelignite, 24 lb. ...		19	3	
Fuse, detonators, lights, etc.		2	0	
Air, maintenance, drills, etc.	1	8	0	= £4 4 3

or a cost of 5s. 3·2d. per ton.

I make the comparison on cost per ton, the only true basis when dealing with the same place. Running three machines per white man brings down the cost per machine, but it is in most cases I know of, a fallacy to think it means cheaper work. Too many machines put on to a man tends to losses in air, machine spares, bad placing of holes and consequently poor efficiency from the machines, needing more machines to break the rock.

As I show in the example, three machines versus two, with three machines the efficiency per white is 21 tons per shift, with two machines it is 18 tons per shift, with costs the same per ton; for the machines, the efficiency per machine is 7 tons and 8 tons, and per person employed in actually breaking the rock, 2·62 tons and 2·66 tons.

The S.A. Mining Journal, a few weeks ago, in criticising the figures of persons employed and tons raised in 1907, mentioned the fact that the efficiency per white had increased greatly, but that the efficiency per person employed had decreased, and I offer the foregoing as a partial explanation.

The rock drilling question is very much like the hoisting question, we provide expensive machinery and do about half the work with it that should be done. I was always taught as a youngster that it was a foolish thing to spend money on machinery without getting the best that it was possible to get out of it. We must look to the efficiency per machine more than we do, working on the bed rock basis of cost per ton.

I saw a long leader in the daily papers a short time ago deploring the amount of casualism in England; what are we doing in this country? Are we not rearing up a large number of casuals? There has been lots of talk about encouraging the married man to settle in the country, and about the evils of keeping the families in other countries, and sending the money out of this country, but what are the married people to do about their children who are growing up? Take the case of the lads; there have been several schemes mooted, one is to take young men of 18 and apprentice them to mining. Prof. Orr and Sir Thomas Price, General Manager C.S.A.R., are looking to engineering apprentices starting at 16; they are right for those people who can afford to keep their sons until they are fit to earn a living at that trade or profession, but there are a very large number of people who really cannot afford to maintain their sons until they are 16 or 18 years old; so what must be done with these? I fancy it is the mines again that must do a large share in the solving of the problem. They are doing a great deal now, and would probably do more if parents would be satisfied to let their sons do honest if menial work.

I would point out to parents that in other countries white boys do all, and girls some, of the jobs I am about to mention; also that they must not let sentiment enter into this matter which is such a serious one for them, for as I have mentioned before, there is nothing wrong in honest work. If they wish to see this country belong to whites they must be prepared to work for it.

The jobs I would recommend for a start, in addition to the blacksmiths, fitters, and carpenters apprentices, are:—Cleaners of engine rooms and boilers, to become firemen and enginemen, office cleaners, or assisting in stores, or at sorting tables, errand boys for shift bosses underground instead of natives, winch drivers in places, coupling trains of cars for haulage motors underground and surface, mule drivers, change houses, and some on drill sorting. There are other jobs which, if little alterations were made, white lads and youths could do instead of running about learning little that is good and much that is bad.

The youngsters working underground would have learned something by the time they were 18 years of age, and would be better fitted to become miners than they would be if only entering the mine at that age.

To heads of departments having charge of any white youngsters I would say, make them work and pay by results. Do not have any of the Kafir servant business with apprentices, they will be better men when grown up by having had to do for themselves. I hope members will consider these suggestions carefully, as so much is depending on this question of white labour here, for it is nearly a matter of life or death for some people. I know there are any amount of people who say white labour is a failure, but I have great reason to doubt it. It takes a great deal to convince some people that higher priced labour can be cheap. The life of machinery should be longer, maintenance less and efficiency per head greater.

It was said that electricity was going to knock out gas, that oil and gas engines were going to scrap steam plants, that turbines would do away with reciprocating engines, but what do we see? The threatened ones are improved and take on a new lease of life. I am of the same opinion as regards the use of more white labour in our mines and other works. Instead of harm being done there will be improvements made, so that on the whole things will be better than before, but we must catch the labour young. I wish others would follow the lead of Mr. Lionel Phillips and the Rand Mines, and run their reduction works by white labour and where possible to go one better on the lines indicated. Members of this Society can do much to help things along by the use of tact and forbearance in handling white unskilled labour. I am sorry to say I have seen bosses who would not use as much forbearance to a fresh white labourer as they did to a similar coloured one.

The President: I would like, on your behalf, to tender a very hearty vote of thanks to the author for his paper. The subject is one that is very much with us in these times, and one which is of very great importance to the inhabitants of Johannesburg and the Witwatersrand generally. I feel sure that the paper will arouse a good deal of interest and, I hope, discussion, and that we will be able to find out more from some of our members on the point as to whether white labour can be profitably employed or not. Of course, one very great difficulty in this matter is getting the white labourer to stick to the job. Continual change of hands is disheartening for everybody, and is not good for the work. With regard to the remarks on the employment of white youngsters in reduction works I think it is very feasible, and it is, of course, very largely done at present. I have some half dozen youngsters under me, and they are working very well indeed. There were a few changes at first, but they very soon found out that the job was not healthy for them if they did not work. I can assure the author that there is nothing of the Kafir servant business on this job. They have to do for themselves, and I do not intend that there shall be any Kafirs to do for them.

Prof. J. A. Wilkinson (*Member of Council*) then read the following:—

REMINISCENCES OF THE EARLY RAND.

(*Read at August Meeting, 1908*).

By M. H. Coombe (Member).

DISCUSSION.

Mr. George A. Darling (*Member*): In adding my contribution to the author's very interesting paper, I may state that I arrived in Johannesburg in August, 1889, and my first experience of mining was received at the old Salisbury G. M. Co.'s battery on the Natal Spruit, and I can quite endorse our President's remarks about praying for a heavy shower of rain to clear away the banked up tailings, We had endless trouble in keeping the tail race clear. I went to this mill as a learner for two months, giving my services in return for any experience or information I might gain. It was rather fortunate for me that the battery was being thoroughly overhauled just at this time. It was shut down for the first three weeks I was there, putting in new mortar boxes, etc., etc., and then I had five weeks with the battery running, and was initiated into the mysteries of gold amalgamation. After this experience I was associated for nearly a year with our Past President Mr. S. H. Pearce, then chief chemist and assayer to the South African Metallurgical Co. In November, 1890, I joined the Gold Recovery Syndicate, and as the Robinson Cyanide Plant was still in course of erection, Mr. W W. Webster (Manager of the Gold Recovery Syndicate) sent me down to the experimental cyanide plant erected at the Salisbury G. M. Co. by the Cassel Gold Extraction Co., and there I obtained instruction in the working of the cyanide process from Mr. Alfred James, who had so ably and satisfactorily conducted all the preliminary experiments prior to working on a commercial scale. While working this experimental plant, I carried out the following experiments and obtained the following results:—

No. 1.—Robinson tailings, $1\frac{1}{2}$ tons treated with 1% cyanide solution, 11 hours percolation and washing.

Before, 16 dwt., 8 gr.
After, 3 dwt., 12 gr. = 78·6% extraction.

No. 2.—Robinson tailings, 2 tons treated with 1% cyanide solution, 13 hours percolation and washing.

Before, 15 dwt., 0 gr.
After, 3 dwt., 0 gr. = 80·0% extraction.

No. 3.—Crown Reef concentrates, 2 tons treated with 0·5% solution, 24 hours percolation and washing.

Before, 2 oz., 10 dwt., 23 gr.
After, 6 dwt., 12 gr. = 87·2% extraction.

No. 4.—Crown Reef concentrates, 2 tons treated with 1% solution, 12 hours percolation and washing.

Before, 2 oz., 10 dwt., 23 gr.
After, 9 dwt., 19 gr. = 80·8% extraction.

No. 5.—Bottle trial (with agitation), 6 hours agitation with 1% cyanide solution. Crown Reef concentrates.

Before, 2 oz., 10 dwt., 23 gr.
After, 5 dwts., 6 gr. = 89·6% extraction.

No. 6.—Bottle trial (with agitation), 12 hours agitation with 0·5% cyanide solution. Crown Reef concentrates.

Before, 2 oz., 10 dwts., 23 gr.
After, 4 dwts., 14 gr. = 91·0% extraction.

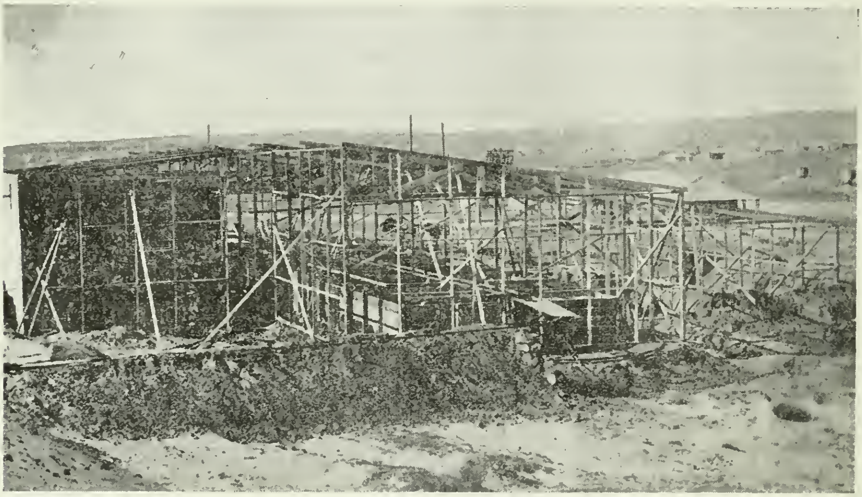
No. 7.—Bottle trial (with agitation), 6 hours agitation with 1% cyanide solution. Sample marked C. B. 816.

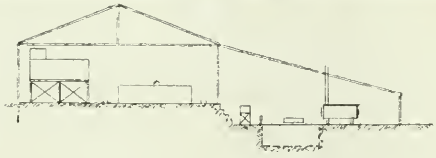
Before, 4 oz., 8 dwt., 4 gr.
After, 9 dwts., 19 gr. = 88·9% extraction.

No. 8.—Bottle trial (with agitation), 6 hours agitation with 0·5% cyanide solution. Gardner Main Reef concentrates.

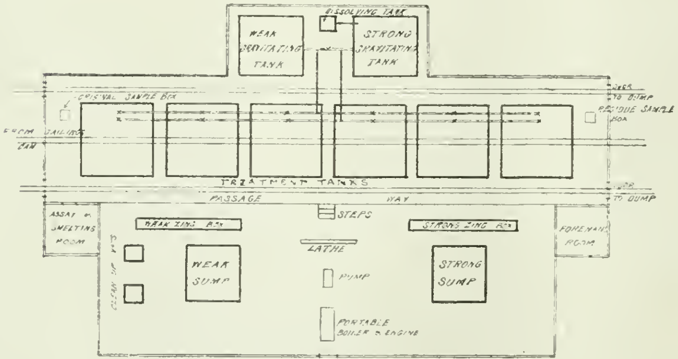
Before, 19 oz., 5 dwt., 11 gr.
After, 11 dwt., 10 gr. = 97% extraction

It was on these and other experiments, carried out during November and December, 1890, that I was able to assure Mr. W. W. Webster that I had every confidence in proving the success of the cyanide process when worked on a commercial scale, and I was particularly anxious to take charge of the Robinson plant when erected, so as to be able to claim that I was the first to prove its success. About the middle of December, Mr. C. McArthur (the then Secretary of the Gold Recovery Syndicate), told me that the Syndicate would like a report on the process from me. This probably meant that they were doubtful whether I had gained sufficient knowledge of the process to run the Robinson plant; at any rate, I told Mr. C. McArthur to say that I would be pleased to write a report, but that my fee for the report would be 50 guineas. I heard no more of this, and did not write any report. I have a photograph of the Robinson plant taken just about this time (before the roof was put on), showing the gravitating tanks and leaching tanks, and rails over the latter for filling. The plant consisted of two gravitating tanks (strong





SECTION



PLAN

and weak solutions), one 400 gallon dissolving tank, six leaching tanks of about 32 tons capacity, two sumps (strong and weak solutions), two zinc boxes, two clean-up vats (v shaped at bottom), zinc lathe, Worthington pump and portable boiler and engine. In passing, I may say that the plant was designed by Mr. Alfred James (to whom a very large amount of credit is due), and I found the two gravitating tanks most useful and convenient, the weak tank for storage, and, no doubt, the strong tank was the means of saving a considerable amount of cyanide, as, when the strong sump showed high in cyanide, it could be pumped out, made up to required strength, and was then ready to be run down on the next charge. On December 23, 1890, the first charge of tailings was put under treatment, and from that day onwards to 30th June, 1891, I was treating Robinson tailings in this plant on behalf of the Gold Recovery Syndicate, and African Gold Recovery Company.

The Gold Recovery Syndicate had contracted with the Robinson Company to treat 10,000 tons of their tailings, paying for the tailings on a sliding scale according to the assay value, and it may surprise many (at least those who have only been here a few years), to know that a clause in the agreement specified that any tailings of the value of *only* 8 dwt. or under, were not to be paid for! The actual tonnage treated during the above dates was 10,485 tons of 2,240 lbs. The first strong solution was intended to be 0·5%, but as a matter of fact, when it was mixed and run on to the charge tested 0·33%, a second dose of solution was afterwards added of 0·45%, and the subsequent strength of solution used in the plant was about 0·5% to 0·6%. My first clean-up (January, 1891), resulted in obtaining 666 oz. of bullion, and there were some long faces, but it stands to reason that a large quantity of gold was returned to the boxes in heavily coated zinc shavings, accounting for the apparent loss or discrepancy, and this has occurred, and must occur, on every new plant which is started, unless the whole of the zinc shavings are taken out and treated. My second clean-up (February, 1891), was much more satisfactory, and as this was possibly the most important month's working in the history of the cyanide process, I purpose giving the Society the actual figures which, to a very large extent, were the means of successfully floating the African Gold Recovery Company in London, registered 10th March, 1891. Up to this time, practically no managers, consulting engineers, or others connected with mining, would believe in the process (with one exception as far as I know), and that was Mr. R. A. Michell, then manager of the Salisbury G. M. Co., but he was conversant with the various experiments carried out at the Salisbury experimental plant by Mr. Alfred James, and afterwards (November and December) by myself. There was not sufficient faith in the process in Johannesburg to float a company. I was constantly asked by "the man in the street" as regards the process, and I always spoke most hopefully of its future, but was much annoyed by the continual remark made that "the gold was half zinc." With the intention of doing away with this idea, I had a small ring made of cyanide gold, and arranged with a jeweller to make it for me for 30s., using only the cyanide gold without alloying. He had a good deal of difficulty in working it, and I had to give him another 10s. for the extra trouble, but from then I wore the ring and showed it to everyone interested. I have the ring at a cost to me of £2, and the Gold Recovery people had the advertisement. It may be handed round for inspection, but I would like it back as a memento of the "early cyanide days." During this important month's working, I made weekly clean-ups (every Sunday), the gold was got into bars as soon as possible, and the result was cabled to London as soon as known.

The following are the most important of the figures for the month's working :—

Tons (2240 lb.) tons.	lbs.	Bullion. ozs.	Average Assay.	Fine Gold. ozs.
257	320	312	730	228
321	960	403	712	287
353	1,280	253	750	190
514	640	606	733	444
1,447		1,574		1,149

In these days bullion ounces were returned by the various companies, and to compare with other returns this output of 1,574 oz. was calculated as equal to 1,285 oz. of Robinson battery gold. The average value of the charges being 1 oz., and the average residue 4 dwt., the theoretical extraction was 80% and the actual gold recovered 79·4%. I may point out that at this time no acid treatment or calcining of the zinc-gold precipitate was carried out. The cyanide (about 65%) used during this month amounted to 2·75 lb. per ton, and the average consumption for the six months was 2·79 lb. per ton, including the cyanide used in making up the first solution. The output consisted of twelve small bars, varying in size from 96 to 305 oz., and in fineness from 704 to 750 fine. I had not been working the process two months before I asked the syndicate to provide me with a filterpress for the clean-up. The result was the arrival at the cyanide works of an ordinary office letterpress, the idea being to fold the zinc-gold slimes in a square of calico and press. Needless to say, the result (in which I took no part) was a fiasco, and I had the satisfaction (?) of seeing a certain amount of my output go beyond all recovery. I will not mention names, but the idea emanated from a high source. During all the time of the treatment of this 10,000 tons the Robinson company's representative had free access to the plant, and the samples of "originals" and "residues" were quartered down in his and my presence every Saturday morning. Weekly samples and tonnage were noted, and the Robinson Co., of course, knew every month the amount of gold obtained. The total recovery from this 10,485 tons of tailings was approximately 8,200 oz. of bullion or 6,000 oz. of fine gold. Theoretical extraction 80%, actual extraction 75%.

The Robinson Co. had the option of taking over the plant (at a valuation) at the end of the contract if satisfied with the results, with permission to treat their tailings on their own account on payment of a lower royalty than charged to others. Mr. Hennen Jennings was then consulting engineer to the Robinson Co., and shortly before the contract ended (June, 1891) he called me up to his office and we had a long conversation. I remember well Mr. Jennings put a very pertinent question to me, he said, "Mr. Darling, you have been treating free-milling tailings, what extraction do you think we shall get from the 'blue'"? Our experience at that time of treating unoxidised ore was rather limited, and, to err on the safe side, I told Mr. Jennings I expected we would get at least 65% of the gold. Mr. Jennings then went into the question as to whether I would continue the treatment on behalf of the Robinson Co., and this being settled, he advised the taking over of the plant, and I was taken over at the same time, becoming an employè of the Robinson Co. as from July 1, 1891.

Some of these facts and figures have appeared before,* when I have had to defend my claim of being the first to prove the success of the cyanide process, *working on a commercial scale*, but by far the greater part has lain dormant for the last 18 years, and only now comes to light.

I hope I have not wearied the Society by going too much into detail.

Mr. C. B. Saner (*Member of Council*) then read the following :—

Mr. A. J. Cotter (*Member*) : The title selected by the author is so very suggestive of the winter fireside with its recognised accompaniment, that I find the flow of reminiscent language to be seriously impeded by the knowledge that it is to be delivered in what, to the lay mind, may be deemed to be in more senses than one, dry surroundings. However, as the chief object is to give members of our Society an opportunity of comparing past and present methods and conditions, I shall endeavour to arrange that the ratio of liquids to solids is such, that no undue silting takes place.

My first experience of a gold mine was in the early part of 1887 when, tiring of banking, I decided to enter upon a field of labour offering greater scope for my then only budding energy and genius. The particular one selected was that of amalgamator on the Beehive G. M. Co., Moodies. In those days, in taking charge of a shift (which by the way was a 12 hours one) previous experience was not considered a *sine qua non*, and the fact that our knowledge, both mechanical and metallurgical, was confined to a few hours instruction, received from our manager, caused neither myself nor my co-shiftsman a moment's disquiet. Why should it? Were we not after a few days, able to strike a tappet key fairly at least once in three attempts, and could we not keep the plates beautifully bright by merely dropping a little cyanide into the mortar boxes?

Further, in determining the percentage of extraction, we were quite independent of that, to us, strange being, an "assayer." After crushing and panning a sample from the feeder, we panned one taken from the bottom of the tables, and if under normal conditions the "tail" from the latter did not exceed ¼ in. we were satisfied that only a portion of the original gold contents was going down the creek.

The mill was a 750 lb. 10 stamp Sandycroft, driven by a Pelton wheel, sunk about 15 ft. below the level of the floor, the whole being covered by a mere skeleton, consisting of a corrugated iron roof supported on rough poles. As the winter at Moodies is sometimes severe, the shiftsmen would under these conditions have suffered from the cold winds. But the construction engineer anticipating this difficulty and knowing that the supply of power water would be irregular, arranged that the shiftsmen should be kept lively. He did this by omitting to instal anything in the nature of a "steady head," with the natural result that at one moment the stamps were engaged in a "wacht een beetje" race, and the next moment they were doing the 100 yards in even time. Under these circumstances, ready access to the main valve was, of course, a matter of consideration, if, as instructed, we were to keep the "drops" as near 75 to the minute as possible. Here again was the engineer's ingenuity in the interests of the shiftsmen exercised, for the valve was so placed that in order to reach it from the tables when "camming" started, it was necessary to take a running jump over the driving belt and skip down a dozen or more steps to a point near the Pelton wheel. It might be suggested that the shiftsmen also should have shown some initiative in their own interests by using the driving belt as a conveyor, when in a hurry to reach the valve. In view of the foregoing, members may be surprised to learn that at the end of six months the mill was closed down and all hands discharged. I then came to Johannesburg; and feeling quite confident of my power of establishing an affinity between copper plates and gold in ore, I applied for, and was appointed to, a position as amalgamator on the Meyer and Charlton G. M. Co., then a 10 stamp proposition. I was not to assume duties for some ten days, and in the interim I casually learnt that they would include driving the mill engine and supervising the firing of the boilers. I therefore called, and explained to the manager that I thought I knew which was the cylinder and which the piston rod of an engine, and with regard to the boilers, I had an idea where the doors were located. He expressed himself as satisfied with my qualifications but suggested that in order to make assurance doubly sure, I should get the man then on shift to indicate to me the lubricating points, and to explain how to cut off steam, etc. In due course I took charge, and all went well until near the close of the shift. Then, happening to touch the piston rod, I found that it was nearly red hot, so I immediately shut down and sent for expert advice. On awakening from my after shift sleep, I was surprised to find beside me, an envelope enclosing 10s.; and a letter immediately severing my connection with the company. I met the manager in town next day, and over a drink and a game of billiards, he laughingly explained how nearly I had succeeded in hanging up the mill for a few days.

---

* See this Society's Proceedings, vol. ii., March, 1897, pp. 28-30.

There was also something original about accounts in those days. The method of allocation was beautifully simple, for the manager sometimes gave it, when making his application to head office for cash. As an example, if he wanted to draw, say, a couple of thousand pounds, he would there and then instruct the office to charge, say, 1,500 to mining, 500 to milling and the balance to general expenses. Even head offices were not free from their little idiosyncrasies. One or two of the secretaries of that time were of a modest and retiring disposition, and rather diffident as to their powers of drawing up an intelligible balance sheet. So, in consideration of a private fee, the company's auditor would come to the rescue. I recollect on one occasion it was found at the end of the financial year, that the company had £600 cash more than the balance sheet called for. Even the auditor could not suggest a reason for this dilemma, so with his and the directors' sanction, it was decided to utilise part of the amount in writing down general expenses, and part in making fully paid up certain of the shares, in the selection of which by the way, an intelligent discrimination was displayed. Two days before the annual meeting, the mystery was explained, for it was found that the "bank balance" had been taken from the "pass book" and that cheques to the value of £600 were outstanding. Of course the above entries had to be removed, and a fresh balance sheet drawn up.

I make no apology for introducing the personal pronoun, for in doing so, I am merely indicating what at the time, was a very general standard of efficiency in the lines touched upon, and also, because it is almost inseparable from a "reminiscence." Nor do I offer an excuse for including Moodies, as it, Barberton, and the Rand; are very closely associated in the memories of the earlier inhabitants of the Rand.

Mr. A. L. Edwards (*Member*): Reading the author's reminiscences of the Rand's early days, one is very sensibly reminded of the great advance that the industry has made.

I note that the author omits all reference to a Natal Board of Directors, who, on being applied to by their manager for funds to extend the shaft, replied by wire; "send measurements of shaft, will have new one made here, and send up by wagon," evidently "shaft" and "disselboom" were synonomous terms in the directorial mind.

Mr. Caldecott, in his contribution, has not called attention to the fact that the patentee of the cyanide process, Mr. J. Stuart McArthur, on his first visit to these fields, was the first to foresee the absolute necessity of a filter press for getting rid of the moisture contents of the ZnAu slimes. I well remember seeing this gentleman experimenting with a letter copying press down at the "Salisbury" Demonstration Plant, then in charge of Mr. Alfred James. The copying press was mounted on a small wooden platform, the whole outfit was then placed in a big bath, the ZnAu slimes contained in a strong canvas bag, were then placed in the press and the screw gently turned, "alas," the effluent was certainly not crystal clear, and its gold contents must have been high, but the interesting result was, a cake of nearly dry slime, which could be handled without any loss.

What wonderful strides the cyanide process has made on these fields since its first introduction here! It is a big jump from the days when 33% pure cyanide, at 2s. 6d. per lb., was sold locally to the first cyaniders, to the present, when 98% cyanide is sold at 8d. per lb., and if the makers can claim merit for this, so on the other hand can the metallurgists of these fields claim recognition of the fact, that by and through experimental work in various cyanide plants, they have been instrumental in reducing the cyanide solution strength from ·75% down to ·1% for sands; whether the limit has yet been reached, it would be rash to prophecy, still it is only a little while ago that Mr. Walford Dowling claimed that the limit of cyanide solution strength had not yet been reached.

Mr. Caldecott recalls the controversy that raged in those early days about the pyritic ore, and it is interesting to note the fact that even then, recourse was being had to fine grinding; side by side with the first cyanide plant at the "Salisbury" old mill site, stood the "Crawford ball mill plant." The patentee of the ball mill was formerly convinced that it was only a matter of a week or two before the cyanide process would give up the ghost, and this sad event was to be brought about by the pre-eminence of the ball mill. Since then, the metallurgists of the Rand have effectually overcome all the difficulties that at one time threatened to lower the efficacy of the cyanide process, and the successful introduction by Mr. Kemmis Betty, of his ZnPb method for effectually precipitating gold from weak cyanide solution, once again rid the industry of the danger of having to pay royalties on the several electrolytic processes then in use, or about to be put in use.

The days of the faddist, of the patent fillers, patent distributors, patent oxidisers, are all past and done with; we now draw our oxygen from an inexhaustible source, and the faddist has successfully been replaced by the trained man and the worker.

The President: If there are any further reminiscences to come from members, I am sure they would be very valuable and very interesting, and we shall be very glad to have them. Of course, the discussion will have to end shortly, but contributions can be printed in the *Journal*.

With regard to Mr. Saner's paper on a "Problem in Shaft Sinking," I may say that the author, when he comes to reply, will have something to add to his paper. He has met with another problem in shaft sinking, somewhat connected with the one he first brought before us, and he will give us that as an addition.

ZINC DUST PRECIPITATION AT CERRO PRIETO.

By Robert Linton (Associate).

(Read at September Meeting, 1908.)

DISCUSSION.

Mr. K. L. Graham (*Member of Council*): In the author's paper the subject of precipitation by zinc dust was dealt with. But, as the solutions used were over ·07% KCN, the success obtained did not entirely disprove the results from the trial run at the Geldenhuis Deep, which I had the pleasure of conducting, in the latter end of 1905, particulars of which are given by Mr. Pearce in his criticism on this paper. Mr. Clark's notes, however, put a different aspect on the subject, he successfully precipitates from solutions containing as low as ·015% KCN, or about the same strength as we run in our slimes plants to-day. In carefully comparing the methods employed both by Mr. Linton and Mr. Clark, with our own, I can only attribute the irregularity of our results to one or other of the following conditions; I place them in what I consider their order of importance:—

1. Agitation with compressed air.
2. Pumping into the press, rather than allowing the solution to flow by gravity.
3. Probable impurity of the zinc dust employed.

Mr. Linton lays stress on the necessity for frequent analyses of the zinc dust, taking particular regard to the quantity of pure oxide and lead present. That the presence of the former would be detrimental, can be easily understood; as to the latter, I notice Mr. Lane Carter states that the Homestake Company stipulates that the zinc dust supplied there should contain 2% of lead.

Another factor which may have to be taken into account when the use of zinc dust is being further considered, which it undoubtedly should be, and which, I think, had a very considerable bearing on the results we obtained, is the amount of white precipitate thrown down in our weak boxes in many cases. This might necessitate the use of a much larger quantity of the precipitant per ton of solution, than either the author or Mr. Clark find necessary. Personally, I should like to thank both the gentlemen for their most useful and interesting papers on a subject that may quite possibly occupy a large share of the Society's consideration in the near future.

The President: I am sure we are all very grateful to Mr. Graham for this addition to the discussion of a very interesting subject.

THE THEORY OF BLASTING WITH HIGH EXPLOSIVES.

(Read at October Meeting, 1908.)

By E. M. Weston, A.S.M.B. (Member).

DISCUSSION.

Mr. Tom Johnson (*Member*): Mr. Weston says he is out for information, and I can assure him that he has no need to feel lonely, for there are a few more of us who would be glad to see it produced.

It is to be hoped that this matter of the use of explosives, or more correctly, the misuse of explosives, will be thrashed out; this Society has many members who could, if they liked, give us quite a large amount of information from their experiences, which would be of great use to the industry. I am afraid some of them have the idea in their heads, that they would be giving away some of their stock-in-trade, but I can assure them that it is impossible to write a paper for our Society, or to join in the discussion on one, without deriving some benefit from doing so, and there is any amount of room for discussion on this subject as with most other underground subjects.

The author is to be greatly complimented on choosing this subject of explosives, and bringing it before us at the present time, for he strikes at one of the glaring instances of waste, one which is as much the fault of the managerial staff, as of the workers.

The cost of explosives per ton mined, is much too high, especially for the amount of rock broken per machine shift. I do not think that many of our deep level mines are breaking ground at under 20 lb. of explosives per fathom, and many are not getting more than half a fathom per blast, which means, say, eight holes per fathom. I am of opinion that in these same deep level mines with a stoping width of 54 in. and over, four holes properly placed and of sufficient depth, say, a 5 ft. bench 8 ft. long, will break about a fathom of ground, and not require more than 20 lb. of explosives, so I do not think the author has brought his paper before us any too soon.

As he points out, economy of explosives can be carried too far, because what we have to look for, is the resultant economy, or economy in total cost per ton broken, in the same stopes. Take for example a stope of 60 in. width; half a fathom per machine shift being broken equals 7·5 tons, machine costs 14s. per shift, say, 2s. per ton, explosives at 20 lb. per fathom, equals 10 lb. If men could be trained to break the fathom with four holes, we could afford to use more explosives if necessary, for 1s. per ton in machine costs would be saved, besides halving the cost of wages for drilling the ground; so that it can be seen that those men who use machines drilling light holes, so as to save explosives, are progressing backwards; also from the above example it can be seen that the mining companies can afford to pay good machinemen more money per fathom than they can pay to poor men, for large cheques are not always a measure of loss to a company, but may be a gain.

In dealing with rock breaking in stopes, I use the rule for beams, with slight modifications. I treat a bench of rock as a beam fast at both ends, if no parting at roof or floor; and fast at two sides, viz., the bottom of the holes, and the side the holes are on. In treating a bench of rock in this way I reason that, first, if I put double the amount of ground on a hole, I shall require to use nearly four times the amount of explosives in the same stope; second, if I lengthen a hole, it does not require a proportionate amount of explosives; as a matter of fact, some holes would break with a less amount of explosives by lengthening them, that is, short heavy holes; often men fail to break a 6 ft. hole that would break with the same explosive if the hole had been 8 ft. deep; third, that the amount of explosives required for a burden of 2 ft. in a 4 ft. stope will break a hole with a burden of 3 ft. in a 8 ft. stope; fourth, that with a smooth parting at roof or floor or both, a heavier burden can be broken with the same explosive than if there were no parting, for the beam is only partially fast at the ends; fifth, that the beam of rock can be broken with less explosives if the bottom end of the bench has room to splinter or tear, instead of breaking off short; all benches should be longer than drilled; sixth, that a bench 8 ft. × 5 ft. of four holes, can be broken with less explosives than a bench 6 ft. × 6 ft. of four holes, the holes being of the usual diameter in use here, for not only is the explosive in a better position to do its work, but the bench is weaker although the amount of rock is greater; seventh, that except in very tight holes gelignite breaks the beam out in better fashion than gelatine will; a hole with 3 ft. burden in a 6 ft. stope is a tight hole in the rock I am used to. I agree with the statement that an 8 in. charge in a $1\frac{1}{2}$ in. hole with a burden of 36 in. is correct, when an 8 in. charge in a 1 in. hole with 24 in. burden is correct. My treatment of this as a beam gives the same result; the weight of explosive is as the square of the diameter of the holes, and the square of the burden is in the same proportion, viz., 2·25 : 1; this holds good for the same place only. The suggestion of the author's that the pressure of the gases escaping along the holes has some rupturing effect is, I fancy, quite correct, for so long as the pressure is greater than the cohesion of the rock, so long will the gases rupture the rock, and as over heavy charges are the rule here, we break the collars of the holes even when the holes are deeper than Messrs. Daw lay down as correct. I am afraid if we were to use only the proper amount of explosives necessary to break the quantity of rock we do, that we should not break the rock, owing to drilling on too short benches, giving wrong direction to the holes, and poor charging. I find the utmost difficulty in getting men to understand how much or how little ground they may have on a hole, and until they know this it is useless to try to get refinement in weight of charge. For this reason men who have proved themselves to be good rock breakers should be put to supervise a certain number of rock drill runners, pointing out the holes and fixing the charges, as proposed in my recent paper, "Notes on Rand Mining." I do not think that the enlargement of holes takes place before the gases commence to move or are condensed.

The reason why a cut comes best when the holes meet, is because of the cone which is thus formed; if it is thought to be due to a pressure surface being formed parallel to the face of the drive, how is it that in so many cuts the front portion of the holes come, and the bottoms are left to be blasted again? I consider the cut is brought by the backward pressure of the gases on the cone, the force of the explosion being straight back and not along the direction of the holes. When the collar of a stoping hole is shot off backwards, as happens now and again, what causes it? The reason is that although there is no pressure surface parallel to the front, the gases exert so great a friction pressure that they grip the rock and tear it off, projecting it backwards in the line of the hole.

The example of economy of proportioning depth and diameter of hole to height of bench of rock is contrary to practice; it is not necessary to use anything like the amount of explosives for the 6 ft. hole; why use $3\frac{1}{2}$ ft. of explosives in this hole? With the $3\frac{1}{2}$ ft. hole there is 70% of the length left for tamping, and in the 6 ft. hole there is 42% of the length left for tamping. I do not consider it at all necessary to use the stated amount of explosives, for if 12 in. in the $3\frac{1}{2}$ ft. hole broke off the other $2\frac{1}{2}$ ft., I think more than $2\frac{1}{2}$ ft. would be broken in the case of the 6 ft. hole, due to the back pressure along the hole; and I think Messrs. Daw are wrong in saying that there must be the same length of vacant hole in each case and, as a matter of fact, in their tables they say differently.

As to increased amount of drilling from higher pressures I plump for longer holes in most stopes, as longer holes are the cheapest way of getting extra rock, and I consider the author deserves credit for his advocacy of long holes. I quite agree with him that it is best to distribute the charge in light holes by interposing tamping to avoid bull-ringing; as to the danger of doing so, I have not noticed it; of course, I have many times seen explosives left in the bottom of holes, but have never noticed it where an extended charge has been used.

There are several causes for explosives being left in holes; not being tamped properly; one hole breaking away the collar of another and taking the fuse and detonator with it; getting the second or later cartridge fast, and forming an air space; front holes being cut out by back holes going first, either through not spitting the front hole or making a mistake in length of fuse. I do not think the author need worry about extended charges being bad practice, it is a very common practice, and is no more dangerous than firing a missed hole through the stemming, which has to be done every day.

I favour No. 8 detonators for machine hole charges, as I think the higher initial detonation is worth at least as much as the difference of price between No. 8 and No. 6.

I do not favour the primer being put on top of the charge, I think it should be nearer the centre of the charge; during my short mining career I have seen many holes charged with the detonator well in the charge without any fear of the explosive "burning," as we use the term, if this is feared, the fuse can be protected. I always like to see a charge well stemmed, and do not believe in the exhortation to "gently press the charge into the hole." I like to make sure there are no air spaces left; the manufacturers of the explosives used in these mines can take credit to themselves, that they produce an article that will stand a great deal more punishment than any man is likely to give it with a wooden tamping stick. The explosive should be tamped down and made to fill the hole, the detonator placed near the centre of the charge, and the tamping tamped down hard, if this were done there would be less accidents from explosives by drilling into old holes. The best diameter to finish off the different classes of holes is as great a problem as any, it depends so much on the nature of the rock, the place, and the burden.

The author gives an example of a hole of 6 ft. depth, $1\frac{1}{4}$ in. diameter, with 3 ft. burden, and allows five sticks of gelatine; now, in the ground that I am used to it would take double that amount of explosive to break that burden; in the one case $1\frac{1}{4}$ diameter is sufficient, in the other it is advantageous to chamber slightly; in the ground I am speaking of, it is good practice to drill the holes about 8 ft. deep, as it only takes about the same amount of explosive to break the burden; I attribute this to the extra length of hole which, allows a greater weight and length on the collar end of the holes, thus retarding the rush of the gases backwards and forcing them to exert greater pressure on the lower part of the holes. In the shorter holes, with this large amount of explosive, the top end of the charge is much nearer the collar of the hole, and on explosion, the gases have a very much shorter distance to travel, and as there is less weight on the collar end there is a socket left: it is noticing these things that makes me agree with the author's idea that the gases escaping along the hole have a rupturing effect. There is as much diversity in the work that the holes have to do, as there is in finger prints, and for that reason any rules that may be got out must be very liberally interpreted. I cannot suggest anything better than to utilise the good rock breakers as supervisors over machine runners.

As to the place for the explosive, I take the expression, "get the explosive to the bottom of the hole," to mean, leave no air space under the charge and do not allow the charge to lie slack.

As to the limit of depth of holes, different mines must decide for themselves: I know that in roofing, or in lifting bottom, 14 ft. holes are very good with from 5 ft. to 8 ft. burden; in 8 ft. stopes 10 ft. or 11 ft. holes would be good, and in 36 in. to 42 in. stopes with machines, say, 6 ft., but with a smooth foot and hanging, then the holes may with advantage be drilled up to 8 ft.; the depth of holes depends a great deal on the ability of the miner to balance them properly.

In machine work especially, economy of explosives is not the only thing to look for: it is not meant that more explosive should be put in a hole than is necessary, but as the author mentions, explosive is only one item of cost, maintenance and labour being much greater.

The determining factor must be, cost per ton broken, so I think we must increase the length and burden, instead of decreasing the diameter of hole. A 48 in. hole will not break as much rock as two 36 in. holes, given the same quality of judgment in planning the holes: with decent judgment, 48 in. holes pay better than 36 in. holes, where the longer holes are suitable. The author shows sketches, and mentions several times, that the rock is broken at an angle of 45° from the hole, or an included angle of 90°; but I find the angle is different for different explosives: with gelatine, we generally get an acute triangle: gelignite, it is more parabolic; gunpowder, more of a segment of a circle; these different forms of break being due to the different speeds of combustion.

In the four-hole bench, I should blast the hole nearest the parting first: with the author's system of drilling it would be the reverse. I should use the same size drills for finishing the holes in the different sized stopes mentioned, balancing the holes to the best of my ability.

The author mentions that if we used a different shaped hole we could save nearly 50% of the explosives; I do not believe it, for several reasons, nor do Messrs. Daw; although they try to prove that it is so, they cover themselves by saying it does not always happen: in the example of the 12 in. × 12 in. × 2 in. and the 12 in. × 12 in. × 1 in. chambers, they try to prove that because there is the same pressure area, the explosive required to fill the latter chamber will do the same amount of work as would double the amount of explosive in the first or larger chamber; they depend on the laws of fluids for their reasoning, but trial is better; the 86% loss of efficiency is forgotten, and it goes without saying, that the loss by escape of gases by holes and fissures, the absorption of heat by the rock, etc., etc., will be practically the same in both chambers, and as P.V. approximately remains constant, P. will be greatest in the chamber carrying the largest amount of explosive, because the volume of gases will not be decreased so quickly; it is to help to prevent the loss of the gases that I advocate that holes should be well tamped.

Messrs. Daw tell us that it would take 12 in of explosive in a hole of 1 in. diameter, 18 in. deep, with 12 in. burden, to break: they also tell us that the same explosive will break a hole over 4 ft. deep, with 3·5 ft. burden; I should not like to be forced to prove it true. I do not think they carried the experiments on ice far enough, having proved what most boilermakers know, that if the circumference of a hole to be punched is increased the pressure necessary is proportionately increased; if they had gone farther, they would have found out something else, that is, if the free face had been increased, the pressure required would have been less; the test with the 4 in. free face gave indications of it, but they did not follow it up, which is to be regretted, for most of us know that greater burdens can be broken by the same quantity of explosives in a high or wide stope than in a narrow one, as the author points out.

In dealing with the pressure of nitroglycerine on detonation, it is given as 12,000 kilos per square centimetre, or about 25,000 lb per square inch, is this a misprint or a mistake? A square centimetre is about $\frac{1}{6}$ of a square inch.

Prof. J. A. Wilkinson (*Member of Council*) then read the following:—

THE GASES RESULTING FROM THE USE OF HIGH EXPLOSIVES.

(Read at November Meeting, 1908.)

By Wm. Cullen (Past-President).

DISCUSSION.

Dr. J. Moir (*Vice-President*): I notice two mistakes, both being the reversal of the formulae CO and CO_2. This occurs (1) in Table II., p. 146, and (2) line 17 of p. 148.

The main point of novelty in the author's paper is the use of very large quantities of explosive in each blast—40 lb. or so as against the 10 or 15

I used in my 1905 experiments.* The author's fumes were thus very concentrated in comparison to mine, so that Dr. Weiskopf's determinations of CO are relatively more accurate than mine, and the ratio of CO to CO_2 is now quite definitely known, instead of being roughly surmised as I had to do in those earlier days.

The few analyses of mine air before blasting here given are very striking, and I wish the author would publish the rest in his possession, also the details of the oxygen determinations obtained in course of this work: there are extremely few oxygen determinations in the Rand mines available, and if we had some more, we might be able to discuss the problem of consumption of oxygen by iron pyrites, which is so striking in Cornwall.

By the way, the author has not as yet found any explanation of the formation of CO and the absence of excess oxygen. May I suggest that the CO is possibly due to the imperfection of the mixture of collodion cotton with nitroglycerine in the explosive; it is known that "cotton" by itself, gives off plenty of CO, and it is easy to see particles of unmixed cotton in all locally made explosives. It is, of course, possible that even when perfectly mixed, the two ingredients explode separately, and give their characteristic gases CO and O_2 side by side, and the temperature of explosion is so high that these cannot combine. As bearing on this we have the author's statement that the wrapper of the explosive is not even burnt when the cartridge is detonated; if this be really so, it becomes quite likely that when blasting gelatine or gelignite is detonated, it is really only the nitroglycerine which explodes and the cotton, the wood fibre and the nitrate of soda are only either burnt or scattered. This curious hypothesis certainly explains the author's new facts all right, and, besides, becomes not so improbable when we consider how extremely sudden the explosion of nitroglycerine itself is.

The other new point in the paper is, that the occurrence of CO in blasting, is not due to bad handling on the part of the miner, but is an intrinsic property of the explosive itself. The care taken in the author's experiments, shows this quite clearly, and I, for one, am glad that the miner has thus been exonerated. Still, as Mr. McArthur Johnston says, this research calls loudly for a reformation in the explosives now in use.

I ought to mention that I have seen Dr. Weiskopf's apparatus, and that it is the completest I have seen, and that—assuming the correctness of the calculations—there is no possibility of serious error in the results. I hope that Dr. Weiskopf's contribution will be ready soon, when these results can be discussed to better advantage.

Finally, I have something personal to say about the author's references to mine ventilation on pp. 148 and 149. I have no desire to gloat over his vindication of my work of 1902 in this field—work done under extreme difficulties, with faulty apparatus such as post-war rubber tubing, and in the absence of essential information about the nature of the samples analysed—but I must say that I think the author might have more definitely referred to the way in which he traduced my work and reputation in 1903. Has he forgotten that he called my work "moonshine," and how the other *illuminati* of the Rand rejoiced over his trouncing of the poor newcomer? I bear no grudge in the matter at all, but I am amused at the author calling on the remains of these *illuminati* to "look at this from a broad, as well as from the individualistic, standpoint." The following quotation is I think quite *apropos* here:—"There are indeed countries in which the average level of intelligence sinks so low, that the inhabitants come to hate, not merely genius, but even talent." Really, it is quite useless to appeal to such, for "a genuine anxiety to get at facts"—they cannot do it—and when such people accuse me of bias or prejudice, I do not care.

Mr. J. Whitehouse (*Associate*): Before making any remarks upon the author's paper, I would like to say that we are greatly indebted to him for bringing forward this most important subject relating to ventilation, and for the valuable information he has given us.

Together with Mr. J. T. Mitchell I have gone carefully through the paper, and we beg to offer a few remarks thereon.

The author's paper has two main objectives:—(1) To demonstrate the impurities existing in mine air as the result of using explosives. (2) To suggest means whereby the formation of these impurities may be reduced to a minimum; thus improving the hygienic conditions, and also the economic working, of our mines. At the same time he naturally refers to ventilation in general.

The paper is confined chiefly to the generation of CO and CO_2, and the ratio of the quantities of these two gases to each other, but the presence of nitrous fumes is also referred to. With regard to the last named gases, he says, "no trace of nitric peroxide was found in any of the samples, though we would not be justified in asserting that it was not formed. I think myself that it must have been formed in every case." With this we agree, and on referring to the paper read by Drs. Macaulay and Irvine in 1906, we find it there stated: "Of the 40 deaths the records points to nitrous fumes as having been the cause of death in 17 cases, in at least 8 of which the diagnosis is definitely stated to have been confirmed by post-mortem examination." Again, we would state that out of a set of 15 samples of air taken from certain mines along the reef, N_2O_4 was reported as present in two of them, whilst it was reported as present in six samples of air taken from faces immediately after blasting.

---

* See this *Journal*, vol. vii., Sept. 1906, p. 70.

Coming now to CO and CO_2, in view of the author's statement concerning the almost universal presence of CO, it may be interesting to note that from 13 samples of mine air in which the CO was determined, 10 samples (77%) gave definite quantities of CO and three gave nil. The following figures may be of interest to members of this Society, and may elicit corresponding figures from others. Of a series of samples taken in certain mines:—

25 gave an average of	...	0·301% CO_2
11 of these 25 gave an average of		0·517% CO_2
13 were analysed for CO and of these		
10 gave an average of	...	0·008% CO
3 gave	...	nil

Now, the author says he holds "rather extreme views" with regard to tamping, and he compares clay tamping with ordinary sand cartridges, and is inclined at present to recommend the use of the former. This matter having been brought forward by him in his evidence before the Commission, we carried out certain tests extending over some three months with special clay tamping. The object aimed at was to get a more perfect explosion with the clay tamping, and consequently, (1) Less vitiation of mine air. (2) More energy directed to doing useful work. We would here state that the test had every advantage in favour of clay tamping as the cartridges were carefully prepared on the surface, and the miners willingly used them, since it saved them trouble and a certain amount of a boy's time, so that the personal antagonism so frequently displayed towards innovations was entirely absent. We failed to obtain the predicted improvement.

In Table II. of the author's paper he quotes figures to support his ideas. We admit the particular figures given there are in his favour, but the percentage of CO_2 in experiment 16b is obviously abnormally low when compared with other samples taken under similar conditions, and it cannot be taken as typical for comparison with 15b.

Taking an average of the experiments quoted in Table I., where ordinary sand cartridges were used, 0·809% CO, and 6·08% of CO_2 are obtained. This brings the ratio of these averages with sand tamping to practically the same figure as that obtained with clay tamping in Table II., and in some cases even better results were obtained. Again, in Table I. no allowance is made for the quantities of these gases present in the air before blasting. If this is done it will be found that in five cases with sand tamping, better results were obtained than with the clay tamping, namely:—

No. 4b	1—10·3
No. 5b	1— 9·0
No. 12b	1— 8·3
No. 13b	1— 9·7
No. 14b	1— 8·7

We venture to express some views on this subject, though they are rather opposed to those of the author. It would seem that in the use of the high explosives with which we have to deal in these mines, what is required is a tamping which will efficiently serve for the momentary confinement of the gases. That we do not depend upon the friction of tamping with the side of the hole is clear, since it is well known that water forms a good tamping. This is due doubtless to the incompressibility of the water, and comparing clay tamping with ordinary sand tamping we reckon the latter is as incompressible as the former. For a lower explosive, possibly, the clay tamping might give better results on account of its closer nature, and therefore lesser permeability.

It has been said, we believe, that on at least one mine a considerable saving has been effected by the use of clay tamping, but failing the production of definite proof of the extent to which such saving is due to clay tamping, we are inclined after due trial to attribute it to other causes—increased efficiency of white and coloured labour. If there be such a saving, it is strange that a larger number of mine managers have not been able to discover it.

With regard to the use of fuse or electric conductors for firing holes, we think that the nature of our blasting operations will make the use of fuse preferable, except in the case of cut holes, for which use alone it may not be considered worth while introducing electric firing. With regard to the fuse itself, however, there is room for improvement.

The author's statement with regard to the blasting gelatine used in experiment 13 is particularly interesting, and we trust his efforts to produce an explosive such as he is aiming at will be crowned with success, both to his benefit and to the benefit of the mine workers.

The question of ventilation is of vital importance to the industry, and this importance is increasing rapidly with the greater depth at which work has to be carried on, and the increased tonnage to be mined. Mine managers doubtless recognise this, as is evident from the various schemes installed and being installed — for example, a Sirocco fan of 300,000 cub. ft. capacity on the East Rand—and these are decided indications that the matter is being tackled. It would seem, however, that the question has not yet generally received sufficient attention. It is only by the systematic circulation of large volumes of air through our mines that we can hope to render noxious gases harmless, and keep the air in such a condition that both skilled and unskilled workers can put forth their best work.

The President: You will remember that when Mr. Cullen read this paper he promised that there would be some other papers on the subject, and amongst these he mentioned one by Dr. Weiskopf. We hope to have that paper read at one of our first meetings at the Transvaal University College, and it is in connection with that paper that the facilities for demonstration will come in, because Dr. Weiskopf I understand wishes to illustrate his remarks by means of a demonstration.

GRADING ASSAYS AND GRINDING EFFICIENCIES.

(*Read at December Meeting, 1908.*)

By Arthur Yates, A.I.M.M. (Corresponding Member of Council.)

DISCUSSION.

Mr. H. A. White (*Member of Council*): The author of this interesting addition to available data on an important and much debated question deserves our heartiest thanks for the completeness of the figures given. It is very evident that a large amount of careful work has been done from which his company should reap considerable benefit in the future in addition to that gained already.

At the very outset it is evident that the ore he is dealing with is vastly different from our average banket inasmuch as a far higher proportion of the gold is inaccessible to cyanide, even in the slimed pulp, and a much larger amount of silver is present. It seems that some of the gold may even be as fine as molecules, and perhaps the author may give us further light on this point. In any case, it is clear that the outstanding difficulty caused by consideration of the finer particles here is greatly aggravated in a case where the difference in extraction on + 200 and − 200 sands amounts to 10%. The commercial estimate of useful work in this case demands the most rigid determinations where they are most uncertain, and a compromise suitable for our conditions would be futile in such a case. This is strikingly exemplified in the July 20th samples. Taking the sum of the efficiency numbers before and after tube mills down to + 200 we find the totals to be exactly the same within limits of error. This means that relative efficiencies have to be based on the − 200 product. Again as difference in extraction on − 200 sands and elutriated slimes is $5\frac{1}{2}$%, it is clear that one separation at least must be made between these products for the purpose of comparing relative efficiency of stamps and tube mills with reference to power consumed. It would be peculiarly interesting at this time if the author would be good enough to give us the comparison of crushing figures, reckoning the − 200 sands, as he suggests, at 0·0025 in., but taking the slimes at 0·001 in. which would, I believe, give a fairer value for the exposed surface.

In this comparison, if the rock be not homogeneous, the fact that the battery gets most credit from the friable and slime yielding minerals cannot be overlooked, as has been done on the Rand in some cases. What goes to the tube mill must evidently have been harder or tougher than the rest of the ore—some little selection has been done in most cases.

From the point of view of extraction, too, in some cases this consideration is of importance. For example, in the case of one ore, the residue of − 90 material after cyaniding the sands from a "700" screen was as high as 1 dwt. If, however, the same sands be all ground to pass as − 90 material the residue is only ·6 dwt., with identical treatment as far as possible. This tends to show that the coarse stuff reground in tube mills to pass a "90 linear" screen will give a lower residue than a similar grade of material yielded direct from the battery. This supports the suggestion of selection in the battery and perhaps indicates why the author cannot use his efficiency factors over periods where milling conditions are changing, and his views on this point would be interesting. It would also be instructive if the author would give us the temperatures of pulp entering and leaving his tube mills as some peculiar results have been obtained on these fields. In certain cases the rise in temperature observed accounted for much more energy than the estimated horse power used, and though suggestions as to oxidation of FeS_2 and rise of temperature due to wetting of larger quartz surface might be made, I believe that it has since been discovered in some instances that the horse power was under calculated.

One lesson is conveyed in powerful form by this paper, which should be taken well to heart in this district. I refer to the results quoted for slimes treated in the sands tanks, and for sand treated in the slimes plant. The first portion of this lesson has over and again been emphasised by our members, and in particular by Mr. J. R.

Williams, and yet to-day many plants, in fact nearly all now at work, partly neglect it.

Recently Mr. Caldecott has arranged some appliances to reduce to the commercial limit the slimes sent in to the sands treatment tanks, and I am convinced, especially where very fine grinding is required, that such methods will be absolutely necessary. The second portion of this lesson should give pause to certain "all sliming" enthusiasts who have not yet comprehended that conservative fact of the difference between "-200" sands and slimes and who so readily take umbrage when this is pointed out.

In conclusion, I would like to congratulate the author of this valuable paper, and our Society upon its choice of a Corresponding Member of Council.

The President: I am reminded that this is our first meeting in 1909. I wish to extend the heartiest good wishes to you all for a very happy and successful year. I may say that the attendance at this meeting is one of the best we have had for some time, and I hope it is a happy augury for the attendances to come.

The meeting then closed.

Contributions and Correspondence.

NOTE ON AN EXPERIMENTAL PROCESS FOR PRECIPITATION OF GOLD FROM CYANIDE SOLUTIONS BY MEANS OF "DECOLORISING CHARCOAL."

As has been shown in a previous note, this variety of carbon takes up 2% of its weight of gold in an alkaline medium, and 4·7% in an acid medium.

Alkaline Treatment.—A vat of wood or of iron, painted with something which will withstand cyanide, of ½ ton or greater capacity, is erected by the cyanide tanks and filled with the gold-bearing solution. From its known contents in gold the calculated quantity of the "decolorising charcoal" (*Entfärbungskohle*, from the Stassfurt Chemical Works, 50 kilos of which have been sent to the Mines Trials Committee) is weighed out into a suitable bucket (taking a slight excess), thoroughly mixed into a paste with a little of the cyanide solution, and the paste thrown into the vat, the bucket being rinsed out with the liquid. The whole is agitated for 20 or 30 minutes when the whole of the gold should be precipitated. The progress of the precipitation should be checked by assaying the liquor.

Acid Treatment.—This is the same except that the cyanide solution is acidified with dilute sulphuric acid in the vat until it just reddens litmus. The advantage is an economy in charcoal.

Separation of Auriferous Charcoal.—This is done by filterpress, turbine, or vacuum-filter. The latter is preferable from simplicity. The pump (actuated by a Körting's steam injector) used in the dye industry is best. It is described in Wurtz' *Dictionnaire de Chimie*, 2nd supplement, p. 1287. The filtered liquid is taken to the storage tanks, neutralised with lime or soda (if acid treatment had been used) and made up with more cyanide as usual.

Should this process prove a success, a methodical treatment using three sets of carbon filters can be installed, so that only the first is saturated with gold and is worked up by incineration, and the fresh charcoal always forms the third filter.

Incineration of Auriferous Carbon.—The moist carbon is spread in thin layers on deep trays of malleable iron, nickel or unglazed earthenware to dry. The waste heat of the muffle is employed for this. It is then incinerated in a muffle similar to that used for porcelain. A cherry-red heat suffices. Great care must be taken to avoid dusting. The brown ash is melted up with borax, sand, and soda as usual. The bullion thus directly obtained is remarkably pure.

Sept., 1908. P. DE WILDE.

MANUFACTURE OF CYANIDE.

In a book called *Complete Practical Treatise on Acids, Alkalis and Salts* (T. Richardson and H. Watts, vol iii., 2nd edition, 1867, p. 61, *et seq.*) I have found a complete description of a process by Possoz and Boissière for making cyanides and ferrocyanides by the action of atmospheric nitrogen on a mixture of wood charcoal and potassium carbonate kept at a very high temperature.* A full description of the furnace used is there given. The process was tried on a large scale at Newcastle by Messrs. Bramwell and Hughes, but owing to unforeseen difficulties had to be abandoned. Nevertheless, in those days it was not known that Siemens' furnaces can produce high temperatures economically, and secondly that the excess carbon from this process can be utilised, viz., as a gold precipitant. This completely changes the aspect of the question, and I think I am not mistaken in saying that by the adoption of such a process the costs *for chemicals* of the cyanide process can be cut down to 3d. Surely this is worthy of serious consideration.

Nov. 26, 1908 P. DE WILDE.

---

* Process tried on the Rand last year.—J.M.

COLOURED WRAPPERS FOR EXPLOSIVES.

Could any of the readers of the *Journal* kindly inform me what objection there would be to having all dynamite and gelatine cartridges done up in red paper wrappers, instead of white, as at present. I am not aware whether this has been considered before, but it appears to me that as quite 80% of the ore being mined is very similar in colour to the white wrappers in use, the provision of red wrappers would to a great extent obviate the overlooking of loose cartridges, which frequently get mixed in with the broken ore, and are a continual source of danger and waste.

Bulawayo, 14th January, 1909. H. T. Brett.

[A copy of the foregoing letter has been sent to a number of consulting engineers for their remarks on Mr. Brett's suggestion, and we have been favoured with the following replies.—Ed. Com.]

I am in receipt of your letter of 26th inst., covering copy of a letter received from Mr. H. T. Brett, Bulawayo, and would state that I consider Mr. Brett's suggestion an excellent one and worthy of adoption: A distinctive coloured wrapper would facilitate the detection of unexploded cartridges in the broken rock underground and much more so on the sorting tables.

W. W. Mein.

Johannesburg, 27th January, 1909.

With reference to your letter of the 26th inst., enclosing copy of a letter from Mr. Brett, regarding the use of coloured paper wrappers for gelatine, I consider that Mr. Brett's suggestion is a good one and well worthy of consideration. The use of coloured wrappers would enable the shovellers underground to easily pick out from the broken rock unexploded cartridges, which, in the existing white wrappers, are almost unnoticeable. These unexploded cartridges are a source of danger, not only to the shovellers themselves, but also to the men employed in the sorting and crushing stations, the mill, and, by reason of their occasionally being found in the coal, in the boiler house also. The best colour which could be adopted for the wrappers naturally would be that which presented the greatest contrast to the rock mined. Thus, in our gold mines, where the rock is, generally speaking, of a greyish colour, red probably would be the best all round colour to adopt. In coal mines, on the other hand, red would not show up well against the coal, and I prefer to use white wrappers.

J. Harry Johns.

Johannesburg, 28th January, 1909.

Referring to Mr. Brett's suggestion *re* use of distinctive wrappers for dynamite cartridges. Anything that tends to reduce the risk of accidents with explosives deserves serious consideration. If this question is referred to manufacturers through proper channels, they will, no doubt, be willing to take it up, and to decide by trial the most effective form of wrapper, whether of coloured or patterned paper. The dynamite cartridge should be as conspicuous as possible, and readily distinguishable from tamping cartridges made up with pink or white newspaper. The careless habit of leaving tamping cartridges about the workings, tends to make the miner less attentive than he ought to be to stray explosive cartridges.

C. B. Kingston.

Johannesburg, 29th January, 1909.

With reference to the cartridge wrapper. This is a question that you should be advised on by the several managers at the mines. I do not think it matters very much whether the wrappers are white or red. The point is to improve supervision and prevent loose cartridges getting mixed in with the broken ore.

Hy. Stuart Martin.

Johannesburg, 30th January, 1909.

THE USE OF THE VACUUM PUMP IN THE CYANIDING OF SAND.

By W. A. Caldecott (Past-President).

The use of the vacuum pump in cyaniding is as old as the first introduction of the cyanide process on these fields by Mr. J. S. MacArthur in 1890,* but of late years the method of vacuum filtration generally has had far more attention paid to it than at any previous period.

Various old sand plants on the Rand were fitted with small vacuum pumps connected with the leaching vats, but this practice for some time past has not been general. At the joint Simmer Deep-Jupiter plant, which has ten 50 ft. sand leaching vats, a large vacuum pump with a capacity of 420 cub. ft. of air per minute was installed, but being somewhat large has since been replaced by one of a quarter the size. The results proved so satisfactory that the practice will now be adopted on other plants, as the cost of installation and of power for operation is small.

In installing a vacuum air pump, it is connected with the upper portion of a receiver, whilst a plunger pump for the solution is connected with the lower portion of the same vessel, or if the nature of the ground permits of it, a pipe from the bottom of the receiver may be carried down to dip below the solution level in a small vat placed about 30 ft. lower. In the latter case solution entering the lower sump by the pipe from the receiver is re-elevated to the boxes by a centrifugal or other pump. No brass fittings should be employed in any of the above pumps.

---

* See this *Journal*, Dec., 1908, vol. ix., p. 205.

Each leaching pipe from the sand vats is connected with the receiver by a short by-pass pipe fitted with a valve, but when leaching is proceeding freely, as in the earlier stages when the surface of the sand charge is still covered with solution, the vacuum pump is not used, and the full flow of solution under the natural action of gravity proceeds direct to the boxes. When, however, the rate of leaching falls off, the connection to the receiver is opened, and the air and solution pumps started. The former maintains a vacuum of, say, 10 in. of mercury, whilst the latter removes solution continuously from the receiver and delivers it to the boxes This operation is continued for an hour or two after the charge is drained, so as to draw down air into all the interstitial spaces in a way which, owing to capillary attraction, is practically impossible in the usually limited time available for sand treatment with gravitation leaching only. Such thorough draining and aeration is carried out three or four times during the treatment of a charge of sands, and with especial thoroughness prior to discharge of the residues. By employing two receivers, strong as well as weak solution can be drained off simultaneously. Sand ready for discharge, and apparently drained thoroughly by gravitation leaching, will on application of the vacuum pump yield a strong flow of solution amounting to several tons per charge, of which the gold contents would otherwise have gone to the residue dump.

The advantages derived from the foregoing practice are fairly apparent, but may be specified as follows:—(*a*) The time required for the passage of a wash through the charge is much reduced; (*b*) owing to the small percentage of moisture, say, 9%, left in the sand after each draining dry and aeration with the vacuum pump, very little gold-bearing solution is left to mix with and raise the value of the next wash, so that the leachings are soon reduced to a minimum value, and a less volume of wash solution effects a more thorough removal of dissolved gold present than by the existing usual practice; (*c*) the amount of moisture discharged with the residues is much reduced, and (*d*) the aeration obtainable during treatment accelerates the dissolving of the gold by supplying oxygen to the solvent solutions. Besides its use in the leaching vats proper, a vacuum pump is of obvious assistance in the speedy removal of water from a collected sand charge, or of solution where sand is settled from a sand-solution pulp, so that the charge is the sooner ready for transfer, and the collecting capacity of a given number of vats is increased.

In the use of the vacuum pump in former times already referred to, the main object was to assist in the leaching of slimy and slowly leaching charges, but under these conditions the solution tended to travel in the path of the least resistance, and the comparative abandonment of the practice was probably due to the slight benefit afforded by small vacuum pumps under such circumstances. At the present time uniform freely leachable charges, free from slime in the interstitial spaces between the sand grains, yield the best results from the application of the vacuum pump.

LIFE OF LINERS.

Mr. J. F. Walker, of the East Rand Proprietary Mines, has kindly furnished me with the following particulars of comparative life of different sorts of tube mill lining on that group.

LOCAL PRODUCT.

	$4\frac{1}{2}$ in.	6 in.
Locality I.	146·62 days.	133·52 days.
	116·0 ,,	
	75·15 ,,	120·03 ,,
	142·52 ,,	
	157·06 ,,	154·81 ,,
	89·55 ,,	
	155·61 ,,	106·18 ,,
	147·68 ,,	
	121·78 ,,	127·46 ,,
	111·35 ,,	
	89·0 ,,	147·60 ,,
	145·25 ,,	
	128·28 ,,	
Locality II.	96 14 days.	74·94 days.
	88·16 ,,	
	87·20 ,,	102·55 ,,
	78·33 ,,	
	111·30 ,,	
	102·88 ,,	
Locality III.	107·76 days.	51·75 days.
		74·70 ,,
	69·57 ,,	100·62 ,,
		60·31 ,,
		72·78 ,,
		89·09 ,,

IMPORTED SILEX $2\frac{1}{2}$ in. life for 20 sets. 54·6 days.
4 in. not tried on this group.

The names of the firms which supplied the material can be ascertained on application to Mr. E. H. Johnson, P. O. Box 134, East Rand.

G. H. STANLEY.

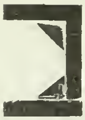
Transvaal University College,
Johannesburg, Feb. 1, 1909.

TABLE OF GRADES FOR LAUNDERS AND PIPES IN REDUCTION PLANTS.

By C. O. Schmitt (Member).

Mill Launders	Battery screen used	mesh	100	200	400	600	1,000
	Grade of launder	%	7½	6	5	4	3
Battery Tables	1¼ in. in 12 in. or 10·4% grade.						
Tube Mill Circuit	Underflow of spitzlutte	10% grade.					
	Shaking table	10% grade.					
	Overflow from shaking tables	10% grade.					
Cyanide Plant	Percentage of +60 in final pulp %		under 10	10 to 20	20 to 40	over 40	
	Overflow of spitzlutte, grade %		3	3¼	3½	4	
	Underflow of slimes spitzkasten (sands pulp) grade %		4	4¼	4½	5	
	Overflow of slimes spitzkasten (slimes pulp)	1¼% grade.					
	Overflow of return sands spitzkasten (slimes pulp)	1% grade.					
	Underflow of return sands spitzkasten (fine sands)	4% grade.					
	Return water launder	1% grade.					
	Leaching pipes from tanks to extractor boxes	1% grade.					
	Solution pipes	1% grade.					

Note.—All grades given apply to straight launders only, and are based on a minimum amount of moisture in the respective pulps consistent with efficient lifting in pumps, or other elevators. All launders should have corner pieces, as per sketch. For curves, an additional 1% should be allowed, and the launder should be lined with rubber. If pipes are used instead of launders, the grade must be at least sufficient to allow the pipe to empty itself without requiring flushing out.



Notices and Abstracts of Articles and Papers.

CHEMISTRY.

A Rapid Method of Qualitative Analysis.—" Walton and Scholtz (*American Chem. Journal*, xxxix., No. 7) have lately shown the value of sodium peroxide as a reagent for opening up ores and other refractory substances, their method being to deflagrate the substances with this reagent and a metallic sulphide. Noyes in his system of qualitative analysis has also used sodium peroxide for decomposing those substances which have escaped attack by acids (*Tech. Quart.*, xvi., No. 2), and for the separation of the metals of the ammonium sulphide group (*Journ. Am. Chem. Soc.*, xxx., No. 4).

The ease and rapidity with which substances are decomposed when deflagrated with sodium peroxide, together with the fact that when the resulting melt is dissolved in water and filtered, a sharp separation of the elements into two groups occurs, has suggested that this treatment might be made the basis of a system of qualitative analysis. The sub-division of each of the groups thus obtained might then be accomplished by means of the ordinary group reagents.

The decomposition of the substance by alkali instead of acid is therefore the essential characteristic of the method.

The advantages of this procedure may be stated as follows:—

1. The method can be applied to practically all substances.
2. Large quantities can be opened up with as much ease as small amounts.
3. All organic matter is destroyed.
4. The group separations are always done under standard conditions.
5. A great economy of time is effected.

It may be argued that a substance like haematite is not completely attacked when deflagrated with sodium peroxide. This would be fatal in quantitative work, but is not a serious drawback when we only require to know what elements are present in the substance; and, moreover, the process can be repeated on the undecomposed portion if desired. The fact that silicates, which constitute the chief portion of the earth's crust, are quite easy to examine in this way, but require special treatment when the ordinary system is followed, is a point strongly in favour of some such modification as the above.

It may also be said that if the substance is not first dissolved in some solvent, much useful information concerning its nature is lost. It seems more systematic, however, to find out first what elements are present, and then to ascertain how they are combined, and to reserve this test till the end of the analysis.

As sodium peroxide attacks all materials it is necessary to arrange the reaction in such a way that metallic impurities are not introduced.

This can be done by making a mixture of approximately one part of the substance, in the state of finest powder, with one part of vaseline and five parts of sodium peroxide, and igniting it on a thick iron plate, or in the cavity of a scorifying mould by means of a match or Bunsen burner. If the metal support is thick the heat is conducted away before the surface can become attacked. The excess of vaseline used also acts as a protection to the metal.

The substance under examination should be dry, and the sodium peroxide used should be quite fresh, as samples which have been exposed to the air deflagrate very badly. Too much vaseline also should not be used, as the energy of the reaction is thereby considerably lessened. As a result of the action of the sodium peroxide, every element now exists in a definite condition in the melt, which is independent of the way in which it was formerly combined.

The melt is now dissolved in water and filtered, when a sharp separation of most of the elements into two groups is obtained. The soluble portion contains the soda salts of the metallic and non-metallic acids, for the most part in their higher states of oxidation, together with those oxides of the metals which are soluble in an alkaline solution. The insoluble portion contains the oxides, peroxides, and carbonates of the metals, any insoluble soda salts, and possibly a portion of the substance which may have escaped attack. A few elements like lead will be found in both portions, but this occasions no difficulty.

It must be noticed, however, that the results obtained by extracting the melt with water are not always the same as those obtained when a solution of the metal is precipitated with sodium peroxide. For example, on extracting the melt, titanium finds it way into the soluble portion, and small amounts of iron also dissolve under certain conditions.

Each of these portions can now be tested with the usual group reagents, but as the groups are smaller the recognition of the individual members becomes easier.

On dissolving the insoluble portion in acids, a little of the undecomposed substance may remain, together with the excess of vaseline, but these may be filtered out. A little phosphoric acid, which has not been rendered soluble by the action of the sodium peroxide, may also be present in the solution.

Instead of testing the soluble portion of the melt with the group reagents in the usual way, the following may often be sufficient:—The solution is first acidified with hydrochloric acid, when the colours due to the formation of perchromic acid and titanium trioxide appear. The solution may then be neutralised with excess of soda, a little sodium sulphide added, and then boiled. On filtering out the metals precipitated by the sulphide, the solution may be again acidified with hydrochloric acid, and the sulphides of those metals, which form thio-salts, filtered out. A little silica may separate here, but this does not in any way interfere with the examination of the sulphides.

After oxidising the filtrate, aluminium may be precipitated by boiling with excess of sodium acetate and a little sodium phosphate. Potassium can then be precipitated in the filtrate as cobalti-nitrite. As, however, potassium is sometimes present as an impurity in sodium peroxide, this fact must not be overlooked when testing for that element.

Sodium must, of course, be looked for in the original substance, and mercury should also be tested for separately on account of its volatile nature.

In examining ores and minerals in the field, it is seldom necessary to test for more than two or three elements; the results, however, must be obtained quickly, and with the aid of a very limited supply of apparatus; in cases such as these the method may possibly prove of use."—W. P. Pollard.—*Chemical News*, Oct. 30, 1908, p. 211. (E. H. C.)

New Method of Making Hydrogen.—"When water is added to the product obtained by treating aluminium filings with powdered mercuric chloride and potassium cyanide, hydrogen is evolved, the reaction being accompanied by a gradual evolution of heat. The heat evolved can be used to regulate the rapidity of the decomposition. This method of preparing hydrogen (which may find application in aeronautics) has the following advantages:—It is very simple, the gas is pure, and the yield of hydrogen is very high for a given weight or volume of the metal."—M. Mauricheau-Beaupré, *Comptes Rendus Hebdomadaires des Séances de l'Académie des Sciences.—Chemical News*, Oct. 2, 1908, p. 173. (E. H. C.)

Determination of Succinic Acid in Presence of Fixed Acids in Wines and Fermented Liquids.—"A known volume of liquid is neutralised by ammonia and acidified with a slight excess of acetic acid, and then barium chloride is added to precipitate oxalic and mineral acids. The liquid is filtered and the filtrate is boiled; excess of lead acetate is then added to precipitate albumenoids, tannins, etc., which are separated off by filtration. The filtrate is treated with sulphuric acid, then with sulphuretted hydrogen, filtered and boiled to expel excess of sulphuretted hydrogen. To separate the malic and succinic acids in the solution it is acidified with sulphuric acid and oxidised with potassium permanganate. The succinic acid remains in the solution, which is boiled; a little potassium bisulphite is added and a large excess of barium chloride to remove the sulphuric acid. The filtered liquid is concentrated, ammonia, alcoholic barium bromide, and three volumes of alcohol are added, and thus barium succinate is precipitated. This is ignited in a muffle, and the barium carbonate determined by titrating with decinormal hydrochloric acid."—E. Pozzi-Escot.—*Chemical News*, Nov. 13, 1908, p. 245. (E. H. C.)

A Characteristic Test for Hippuric Acid.—"It can be applied directly to urine, and yields definite results in solutions so dilute even as hundredth-normal. A few cubic centimetres of the urine in a test-tube are first treated with a quantity of sodium hypobromite solution, just sufficient to decompose urea and to impart to the mixture a permanent yellow colour. The solution is then heated to boiling; an orange or brown-red precipitate is formed, if hipparic acid is present. When present only as a trace the solution appears smoky and faintly red in colour; when present in larger quantities the solution is rendered opaque and orange or brown-red in colour. In either case after standing for some time the solution clears up and a light, finely divided precipitate settles. It consists of white 'earthy phosphates' intermixed with the amorphous orange or brown-red solid about to be described. (Application of the hypobromite test to aqueous solutions of other constituents of urine gave no colour effects. The following were tested:—Uric, benzoic, oxalic, and fatty acids, creatinine, acetone, acetacetic ester, glucose, glycogen, and leucine.)"—W. M. Dean, *Journal of the American Chemical Society*, xxx., No. 9.—*Chemical News*, Oct. 9, 1908, p. 181. (E. H. C.)

The Production of Ammonia from Atmospheric Nitrogen by Means of Peat.—"Attempts to utilise the nitrogen contents of peat have been made during the last sixty years, but the yield of ammonia obtained by the various processes which have been tried shows that only about one-third of the nitrogen can be thus recovered. Starting from an observation that a mixture of nitrogen and hydrogen passed over reduced iron at a low heat always produced ammonia for some time, the author made further experiments on these lines, and found that the presence of oxygen and water was of importance. This led to the gradual reduction of the quantity of hydrogen, and finally to its complete omission; but the periodical necessity of reducing the iron used for the purposes of moist oxidation led him to employ some carbonaceous material (coke, coal, charcoal, etc.), for this purpose.

Finally, peat was found to give the most satisfactory and rapid results, and even when it contained up to 80% of water it could be utilised advantageously.

In the meantime experiments were made to prove the co-operation of atmospheric nitrogen. Sugar carbon absolutely free from nitrogen was treated by the process, and produced an average of 1% ammonia on the carbon consumed.

The large plant erected at Carnlough, Co. Antrim, has finally confirmed the working and results of the process on a commercial scale."—H. C. Woltereck, British Association Dublin Meeting, 1908.—*Chemical News*, Oct. 2, 1908, p. 163. (E. H. C.)

Metallic Reduction Process.—"A new company has been formed in France for manufacture of metallic oxides by a very original process, though only applicable for metals with a low melting point, like zinc, tin, and lead. The molten metal falls in a thin stream, and is pulverised by violent impact with a jet of steam at 2·8 atmospheres. The impalpable powder thus produced is immediately bathed in a current of cold water, screened, collected, and dried. The metallic particles thus obtained pass rapidly, under the action of a strong blast of air, through the flame of a blow-pipe, to be transformed into oxide and then drawn by suction into condensation chambers. All the operations are performed in closed compartments so that no metal dust or oxide can float about in the air of the workshop. This new invention is particularly interesting from the point of view of hygiene in manufacture of lead."—*Revista Minera l'Industrial de Linares*, Oct. 20.—*London Mining Journal*, Nov. 7, 1908, p. 577. (A. R.)

METALLURGY.

Automatic Filter-Press.—"The authors have just completed the working model of a rotary filter-press. Upon twelve rollers a table, 16 ft. in diameter, revolves, being driven from a tappet wheel, which is in gear for half-revolution. When in gear it works in cogs round the circumference of the table, and when free it fills another purpose, that of acting as a break on the table, and holding it firm. From a powerful skeleton over the table are suspended 10 pinioned wheels whose eccentrics, by their lift and fall, press anything that comes beneath them. The principle of the invention is that the table, which is divided into 12 circular compartments for holding material to be treated, shall revolve by a kind of ratchet movement, and be clamped tight for the moment it is at rest. It is designed so that the tailings shall be fed from overhead, and that each compartment, as it comes under the feeding pipe, shall touch a lever which will allow a given quantity to fall into the vat. The tappet wheel coming into gear moves the table round, and then clamps it as before. This operation brings an empty vat under the feed pipe, and moves the one which has just been filled under a pinion wheel, geared so that it works a disc with hollow teeth with a circular motion through the stuff, and thoroughly disintegrates it. Water is forced through the teeth, and so assists the agitation. The next move of the table brings the agitated stuff under one of the presses, where an arm from the crank of one of the moving pinion wheels exerts a pressure upon it of 3,000 lb. The vat and press head are tight fitting, and there is a rubber lip on the vat which ensures it being airtight the moment the press descends and prevents any of the stuff oozing upwards. It thus forms an air cushion between the tailings and the press head. The geared wheel which works the press is kept down by strong springs, which ensure a steady force and at the same time minimise the risk of a breakdown. A pressure of 3,000 lb., the inventors claim to be more than sufficient to force the pulp through the cloths, and the cyanide liquid with the gold in solution is forced through the perforated bottom of the vats into a launder which encircles the table and flows into whatever receptacle may be designed to store it pending treatment. The pressure under each individual press only lasts a few seconds—the while the tappet wheel which moves the table is making a half revolution, and coming into gear again. In this way the vat of tailings goes the round of the table, during which time it is twice washed and agitated with a revolving disc with hollow teeth, through which water is forced, and eight times pressed. The liquid escapes through a filter cloth. When it comes to within one compartment of where it was filled, a lever underneath the table lifts the bottom of the vat flush with the table, and an arm worked from a shaft pushes the solid cake of tailings over the edge of the table on to the conveyor or whatever contrivance may be designed for the removal and stacking of the residues. The table revolves once in 40 seconds, and during that time each compartment is automatically filled, subjected to two washings, and agitations, and eight pressings before being lifted up and pushed over the table. One man can easily attend to the machine, as all he has to do is to see that the bearings are kept oiled. The estimated capacity of the press is 40 tons per hour, and it is estimated that a 15 h.p. engine can drive it. It should not be an expensive machine to build, as the major portion of it will consist of angle and channel iron, such as is kept in stock on all mines, and which will only require to be bent and shaped to fit. The cogged wheel round the circumference of the table is designed to be put together in segments, so that should a tooth snap the segment can be unbolted and a fresh one put in with very little loss of time. The inventors have had it protected under the name of the Lemon-Holmes automatic filter-press."—R. D. Lemon and R. P. Holmes.—*Australian Mining Standard*, Oct. 28, 1908, p. 483. (A. R.)

Polishing Metals for the Microscope.—"To polish metals for examination under the microscope Mr. Robin employs alumina, which he prepares as follows:—Pure aluminium (commercial aluminium is very suitable) is rolled rather thin and has a somewhat great natural polish. The metal is then cut into bands 5 mm. wide, and a bundle of them placed in a bottle containing a few cubic centimetres of mercury, which is well stoppered. After shaking well for two minutes the aluminium bands are taken out and placed in a crystalliser exposed to the air, or in a large glass of water covered over to keep out dust. The same mercury can be used again for other operations. Aluminium thus treated when left in the air becomes covered with white tufts of alumina, which visibly enlarge. This operation, which seems more productive in damp weather, ceases in four hours or so. Work can be repeated until all the aluminium bands are exhausted. The alumina produced, very tenuous and light, seems to give a somewhat greater weight than that corresponding to formation of $Al_2O_3 3H_2O$ (the alumina absorbs moisture from the air so that the process is not successful in dry air or oxygen). It can be immediately employed for polishing that is to say, in hydrated form. Experiments with this polishing material were made at the Laboratorie d'Essais du Conservatoire des Arts et Métiers. This process may be useful to metallurgists and micrographers. Another method to treat the aluminium bands is also described in *Revue de la Metallurgie* of October."—Mr. ROBIN.—*London Mining Journal*, Nov. 21, 1908, p. 648. (A. R.)

PRODUCTION OF THE RARE METALS.—"The *Revue de Metallurgie* gives an epitome from an article by Mr. Walter Giessen of the actual industrial conditions of some rare minerals. Thorium, for example, utilised for gas mantles is found in Brazil and small quantities in the United States. The latter and Australia supply tantalum (tantalite and colombite), which is in demand for Siemens and Halske electric lamps. Zirconium extracted in form of oxide and gadolinite is also employed for electric lamps (Wedding and Nernst). Vanadium, chiefly from the United States and Mexico, improves the steel utilised in France for chassis of automobiles. Uranium, as we are aware, is from Joachimsthal. Molybdenum is now extensively used for quick-cutting steels or weapons; it facilitates the drawing out of nickel and gives the wire greater resistance. It is found in Australia, where Queensland can supply 20 tons of ore annually, Canada 70 to 80. There are also deposits in Scandinavia. The price is still high, about 8·25 frs. per kilogramme for a 95% ore."—W. GIESSEN.—*London Mining Journal*, Nov. 14, 1908, p. 617. (A. R.)

MINING.

MINE VENTILATION.—"In the Amending Mines Act (Victoria), passed on December 23 last, several of the old regulations regarding ventilation were amended or supplemented. For all mines except coal mines the minimum quantity of air per man per minute is 70 cub. ft., and a maximum of 500 cub. ft. where noxious gases or high temperatures prevail is adopted as the standard of quantity. Carbon dioxide is fixed at 25 parts in 10,000, instead of 30 parts in 10,000 as previously. It will be remembered that the West Australian Commission recommended 15 parts in 10,000 of air by volume as a standard of quality. The standard for oxygen remains the same. 'The amount of oxygen in the working places or the approaches thereto shall not be less than 20% by volume. The maximum temperature of the air in any working place shall not exceed 83° F. by the wet bulb thermometer.' The Chief Inspector of Mines says that the air in the deep mines of Bendigo usually contains a high percentage of moisture, and is either saturated (both wet and dry bulb thermometers read the same) or the dry bulb reads 1° or 2° higher than the wet bulb—1° equalling 95% of moisture, and 2° 90% of moisture approximately. Mr. A. H. Merrin (Chief Inspector) further says: The standard of carbon dioxide but little affects the quartz mines of the State, as in such mines, shallow or deep, the carbon dioxide is rarely greater than one-half the amount fixed as permissible. The standard has to be fixed with regard to the deep alluvial and coal mines. In the majority of the working places in such mines as the last the carbon dioxide is under the amount adopted, and where the quantity of air per man is supplied at 70 cub. ft. per minute (the minimum quantity), CO_2 rarely reaches to the new standard. If the Chief Inspector is satisfied that from any cause it is not practicable to maintain the air at such a standard as is prescribed in the Act, he may determine the hours to be worked (not exceeding six) by any person in each working face.

The Inspector may also order connections to be made between levels of adjoining mines, or the use of auxiliary ventilating appliances. All 'centre country' winzes or others used for ventilating purposes must be sunk from a chamber cut in the side of a drive.

Where a new shaft is started, and to be sunk below 600 ft. from the surface, one compartment shall be close timbered and have an area not less than 10 sq. ft., and be used solely for ventilating purposes. The depth of 600 ft. expressly excludes alluvial mines or prospecting mines from the operation of this regulation. It is stipulated that the fourth compartment shall be placed at one end of an oblong shaft, and the ladder way at the other end of the shaft, with the winding compartments between them, so that the air current may be carried down the shaft or returned up the shaft, free from obstructions.

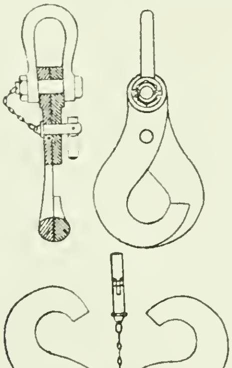
In Bendigo mines at a vertical depth of 4,000 ft. water issues from the rocks at a temperature of 114° F., and the temperature due to the heat of rocks is 110° F. But the actual temperature at this depth will depend upon the quantity of air current and the time it has been circulating to drain off the rock heat.

In summarising the Amended Regulations I should have mentioned that managers are required to keep plans showing the direction of air currents, and the positions of air ways in their mines.

Regarding the tests for CO or CO_2 which are made by inspectors of mines infrequently, I do not think anyone takes such tests now as indicating anything more than an approximation, unless one shows a very high percentage of such gases. Our Mines Department furnishes its officers with a closed tube containing a standard solution of barium hydrate, with added phenolphthalein, and a tube containing two valves and a rubber press ball attached. The pink coloured solution of the first tube is transferred to the last by breaking it carefully in the last. Being closed, it is shaken, and air bubbles forced through by single squeezes of the press ball until the pink colour disappears. Notes of the air replacements made by squeezing are taken and compared with a table in order to estimate the percentage present of CO_2."—*London Mining Journal*, Oct. 31, 1908, p. 542. (A. R.)

NON-SPINNING ROPES.—"It is reported that a non-spinning rope has been recently made for use in quarries, mines, and similar places, where it is desired to hoist heavy weights with a single rope. With the use of the rope of standard design, the load has a tendency to rotate, because the action of the weight is to untwist the strands, and this tendency must be overcome by the use of a guide rope, by which the load is controlled on its journey through the air. As a test, a block of marble suspended by the single line was raised from a depth of more than 250 ft. without making a half-turn in its ascent. No guide rope was necessary. In the construction of this rope an inner series of wire strands is wound in one direction over a core and an outer series of wire strands is wound in the opposite direction about the inner series as a core. The wires which make up each of the individual inner strands are twisted in the same direction as the lay of strands, but in the outer strands the individual wires are laid in opposite directions to the lay of the strands. The number of strands required for the second or outer layer must, of course, be greater than the number of strands in the inner layer. For this reason the inner layer has the wires as well as the strands all laid in one direction as described, thereby giving them a tendency to untwist very much faster than if they were twisted in the direction opposite to the lay of the strands. The wires in the outer layer, being laid in an opposite direction to the lay of the strands, do not tend to untwist as fast as the inner layer. The strands in the outer layer, however, counteract the tendency of untwisting the rope as a whole. When a weight is applied to the end of the rope the greater number of strands of the outer layer have a tendency to untwist in one direction, whereas the few strands of the inner layer have a tendency to untwist very much faster in the opposite direction, and it is found in practice that these two actions almost perfectly balance each other."—*Science and Art of Mining*, Dec. 12, 1908, p. 207. (A. R.)

A Safety Hook for Shaft Sinking.—This sketch shows a hook designed by Mr. T. H. Prisk, of New York, to meet special requirements in connection with shaft sinking. It has no springs or other delicate parts to get out of order. It is, the designer says, impossible for the bucket to drop off the hook. It is easily attached and detached, and is so simple that it can be made by an ordinary smith.—*Melbourne Leader*, Oct. 10, 1908. (C. B. S.)



Record Speed in Drilling.—"A borehole put down near Dorsten by a Duisburg firm of contractors has struck the coal deposits at a depth of 1,087 metres in sixty-five days, including a stoppage of fourteen days, caused by the difficulties encountered in proceeding with the work."—*London Mining Journal*, Nov. 21, 1908, p. 647. (A. R.)

Coal Dust in Mines.—"The coal dust experiments at the Altofts colliery were continued during last week, Yorkshire and Warwickshire being represented by a party of about 80 visitors concerned in mining. The demonstrations have so far proved eminently successful. According to Mr. W. E. Garforth, who is conducting the experiments, it has been made abundantly clear that coal dust possesses both inflammability and explosive force. He expressed the opinion that the use of water as suggested by the Home Office would prove very disastrous to the coal owner. Stone dust, as demonstrated by the recent experiments, was equally efficacious. It was hoped, he added, that as a special result of these experiments, 90% of deaths from explosions in coal mines, on the basis of the past returns, would be avoided."—W. E. Garforth.—*Times Engineering Supplement*, Nov. 4, 1908. (J. A. W.)

Mining Accidents in the United States.—"In consequence of the number of mining accidents in the United States legislative action is being considered, and in Pennsylvania a Bill has been prepared by the provisions of which a small tax is to be placed upon every ton of coal sold from Pennsylvania, the proceeds to form a fund from which the victims of mine accidents or their surviving families will benefit. It is held that the tax need only be $\frac{3}{4}$ cent per ton on bituminous coal, and $1\frac{1}{2}$ cents a ton on anthracite coal."—*Iron and Coal Trades Review*, Nov. 27, 1908, p. 2,313. (A. R.)

Mine Explosions.—'The recent disastrous coal mine explosions have caused much discussion as to their causes, and as to what can be done to prevent their recurrence. I beg leave to make a plain statement of a number of the known causes that have come to my knowledge through my experience as an inspector.

It is an acknowledged fact that the older the mine, the more dangerous it is. This is especially true where mines penetrate the earth to a considerable depth, and it does not matter whether the coal is reached by drift, slope, or shaft opening, the results are the same. It is also known that some mines are more dangerous than others, because of the quantity of gas generated and the deposits of coal dust therein. It does not matter how small a quantity of explosive gas a mine may produce; experience has taught that mines generating even small quantities can have explosions therein that result in great loss of life and property. This is particularly the case where coal dust is also produced in dangerous quantities.

That coal dust has become an element of danger in mines cannot be denied. Under ordinary conditions the amount of dust in suspension in the air is not sufficient to produce an explosion. A comparatively small explosion of gas or a blown-out shot accompanied by a flame and of sufficient force to start up a dense volume of dust in its immediate vicinity, is the usual cause of coal dust explosions. The dust, having been ignited, is carried along with increasing intensity and velocity as long as there is sufficient dry dust and air present or as long as the necessary conditions exist.

Many experts maintain that most of the general and destructive mine explosions have been due mainly or entirely to coal dust and that the explosions of gas were only incidental and local. This appears to be true in many cases from the fact that many explosions of wide extent have taken place in mines where very little if any gas had been known, and which were well ventilated.

Where a mine is well ventilated there is less danger of an accumulation of explosive gas, as the gas would be diluted and carried away as fast as it was given off by the mine. Of course, 'blowers' or 'feeders' may be encountered of sufficient strength to flood a portion of a mine with gas and its ignition would cause much damage.

It is a fact that explosions have followed all main air-ways and along the main haulage roads in the best ventilated parts of a mine, and have avoided the return air-ways where the air would naturally contain the most gas. This can be accounted for by the dust theory. The main haulage ways are naturally dusty and there is plenty of air to support its combustion after ignition.

The question is often asked — 'Why do these explosions occur in the mines more frequently in winter than in summer, and why are they more destructive in the former season than in the latter?' In summer the air outside of the mine is much warmer than the roof, sides and floor of the mine. The ventilating current is cooled on entering the mine, rendering the temperature of the air too low to carry its previous charge of vapour in suspension. Condensation then takes place, and the air deposits its excessive charge of water on the roof, sides and floor, and the roads become wet. Whereas, in winter when the ventilating current entering the mine is colder than the roof, sides and floor of the mine, the air becomes warmer and consequently undersaturated, and the result is that the warm air absorbs the moisture from the roof, sides and floor, and dries them so greatly that the coal dust adheres to them and becomes a dangerous factor, particularly so when the air current contains a small percentage of explosive gas. The percentage may be so small that it cannot be detected by the ordinary safety lamp. Coal dust itself is a dangerous element and may be the sole cause of an explosion when acted upon by a flame of sufficient intensity and magnitude, such as that produced by a blown out shot or local explosion of gas. The action of the flame distils carbon-monoxide gas from the fine particles of dust sustained in the air. The explosion of the gas thus formed causes a further disturbance of the air and raises a larger quantity of dust which likewise explodes and liberates fresh quantities of gas, and thus the explosion gains force and magnitude and is transmitted practically to all parts of the mine. The most dangerous explosive elements we now have to contend with in mines are explosive gas and coal dust, either of which may be harmless of itself, but under certain conditions and when brought into contact with flame will cause an explosion. This contact may arise from the following causes, namely: (1) an open light; (2) a defective safety lamp; (3) the flame from a shot; (4) the accidental explosion of highly inflammable explosives; (5) the sparks or flame from electric wires; (6) the flame from any burning material.

The writer, after a varied experience of several years, is firm in the opinion, even where either or both of these dangerous elements exist, and where the agencies of flame above mentioned are in use, such disasters as have recently occurred would be a thing of the past if the following safeguards were introduced and properly maintained:—

1.—Render harmless all issues of explosive gas by constant and sufficient ventilation.

2.—Employ a sufficient number of fire bosses to make careful examinations without undue haste.

3.—Exclude all open lights. Instal locked safety lamps and keep them in good and safe repair at all times.

4.—Employ competent shot firers to prepare, charge, and fire shots after the workmen are out of the mine.

5.—Use the best flameless explosives for all blasting purposes.

6.—Avoid placing electric wires in return air-ways where explosive gas is likely to be carried.

7.—Where coal-cutting machines are in use adopt the one that produces the least dust.

8.—Instal a water system for the purpose of thoroughly wetting and laying the dust; remove all accumulations of dust from the mine after it has been saturated with water.

9.—Allow no timber or other material of a highly inflammable character to accumulate in any part of the mine.

10.—Employ the most competent men at all work involving risk and responsibility.

11.—Enforce rigid discipline and strict compliance with all rules and regulations."—C. B. Ross.—*Science and Art of Mining*, Dec. 12, 1908, p. 205. (A. R.)

Twenty Years' Progress in Explosives.—"At a Society of Arts meeting, held last November, Mr. Oscar Guttman delivered the second of four lectures on progress in explosives. The lecturer said that in the manufacture of nitro-glycerine the operation of nitration was latterly more efficiently controlled, and larger plants were now being installed. In one United States Works four steel nitrators, each for a charge of 1,000 lb. of glycerine, had been placed in one room and driven from one main shaft. With regard to the composition of the nitrating mixture, it was now customary to add sulphuric acid containing 20% of anhydride to the original mixture, and the use of anhydride had reduced the quantity of sulphuric acid required. It depended, however, upon the price of the sulphuric anhydride whether it was advantageous to use it. The separation of nitro-glycerine from the waste acids had been improved by the addition of high-boiling paraffins in quantities of 0·5 to 2% of the weight of glycerine, and Dr. L. F. Reese added sodium fluoride in the proportions of 0·002% of the glycerine employed to the nitrating mixture with excellent results. A number of useful processes, including the displacement process in operation at Waltham Abbey, had been introduced with the object of obtaining a better yield of nitro-glycerine. There had been no special improvement in the manufacture of dynamite since Nobel invented blasting gelatine in 1875, but there had been within recent years a revival of old ideas with some success for obviating one of the chief objections to dynamite, that of freezing. One method was the addition of dinitroglycerine to trinitroglycerine explosives, and he believed that recently dinitrodichlorhydrine had been used with considerable success at the German works of the Nobel company. With regard to gun-cotton, the really important step in its manufacture was taken when the British Government adopted a process of pulping and purifying the gun-cotton. It was very curious that while in the purchase and use of nitric and sulphuric acid for the nitration of gun-cotton most stringent regulations were laid down with regard to freedom from mineral matter, chlorine, sulphates, and arsenic, yet as far as he could ascertain, no special precautions appeared to be taken in the case of cotton to guard against impurities, and apparently it had never occurred to anybody to ascertain whether the impurities in the cotton were responsible for a good deal of the instability of certain finished gun-cotton and smokeless powders. He was convinced that such was the case. He was of opinion that if it was necessary to use nitrocellulose, and if, as seemed to be the case, cotton was the best material for making it, the natural cotton only, and not common yarn, and less still waste, should be employed. It was agreed on all sides that the treatment to which gun-cotton was subjected in the course of stabilisation had a great deal to do with its ultimate composition, and undoubtedly the manufacturer of gun-cotton and nitro-cellulose was face to face with great difficulties. It was recognised, however, that the Waltham Abbey gun-cotton as at present made was a very stable and good article. This was due to the introduction by Dr. Robertson of improvements in the method of boiling, and Dr. Robertson's was one of the most useful pieces of work accomplished since the invention of gun-cotton. Nitro-starch had recently been utilised in the manufacture of smokeless powders."—O. Guttman.—*Science and Art of Mining*, Dec. 12, 1908, p. 194. (A. R.)

MISCELLANEOUS.

Fireproofing Wood.—"Sulphate of aluminium, when raised to a high temperature, gives an infusible residue, a non-conductor of heat which protects the cells of wood. Consequently, it can be employed to make wood fireproof. Not only does it prevent production and propagation of flames, but even the incandescence of wood."—*La Nature*, Oct. 24.—*London Mining Journal*, Nov. 14, 1908, p. 616. (A. R.)

Conservation of Iron Ore and Coal Resources.—"I have for many years been impressed with the steady depletion of our iron ore supply. It is staggering to learn that our once supposed ample supply of rich ores can hardly outlast the generation now appearing, leaving only the leaner ores for the later years of the century. We must in all possible ways lessen the demand upon it, for it is with iron ore we are least adequately provided. One of the chief uses of this metal is connected with transportation, mainly by rail. Moving 1,000 tons of heavy freight by rail requires an 80-ton locomotive and 25 20-ton steel cars (each of 40-ton capacity), or 580 tons of iron and steel, with an average of, say, ten miles of double track with 90-lb. rails, or 317 tons additional, so that, including switches, frogs, fish-plates, spikes, and other incidentals, the carriage requires the use of an equal weight of metal. The same freight may be moved by water by means of 100 to 250 tons of metal, so that the substitution of water-carriage for rail-carriage would reduce the consumption of iron by three-fourths to seven-eighths in this department. At the same time the consumption of coal per motive power would be reduced 50% to 75%, with a corresponding reduction in the coal required for smelting.

The next great use of iron is in construction, especially of buildings and bridges. Fortunately the use of concrete, simple and reinforced, is already reducing the consumption of structural steel. The materials for cement and concrete abound; and while the arts of making and using them are still in their infancy the products promise to become superior to steel and stone in strength, durability, convenience, and economy of use. The cement industry is growing rapidly, largely in connection with the making of iron and steel, so that the substitution of the new material will not involve the abandonment of plants or loss of invested capital. The hitherto useless slag hills, of which many may be seen around blast furnaces, are now being made directly into cement and yielding high profits. It has become a bye-product, the extra cost being scarcely more than the former cost of piling the slag away.

A promising mode of reducing iron consumption is opening through the development of iron alloys. Among the most abundant materials of the earth-crust are silica, alumina, and carbon compounds, all with more or less affinity for iron; already the alloying of carbon with iron has revolutionised the industrial world, and of late the alloying of silicon with iron (in 'ferro-silicon,' etc.), gives promise not only of yielding a superior metal, but of permitting the reduction of silicious ores hitherto unworkable, while aluminium has been alloyed with iron in a useful way. It is not too much to hope that research into the ultimate constitution and relation of these common materials will yield both better and cheaper metals than any thus far produced, and that newly discovered alloys will help to relieve the pressure on our mines of iron, copper, zinc, silver, and lead.

Coal.—The most serious waste of coal arises from imperfect combustion in furnace and firebox. The waste of 90%, and over, of the potential energy of the fuel in power-production—which, however, we know not yet how to avoid—is appalling in itself, while the smoke and soot from the chimneys becloud and befoul cities, poison human lungs, and prepare the way for pneumonia (one of our worst modern scourges), and initiate all manner of additional wastes. We have already learned that internal-combustion engines and gas producers double or triple the power per unit of coal, obviate the smoke nuisance, and also permit the use of lignite, culm, slack, and inferior coals—in fact, so far as power-production by reciprocal engines is concerned, the days of steam seem to be numbered, although the development of substitutes is still in its infancy. The consumption of coal in smelting is necessarily large; of late the loss is reduced by using the furnace gases for power and by making bye-products; yet the chief saving must lie in economy in the use of metals. Much of our coke-making is still extravagant; some ovens use the gases; and all should do so without delay—if necessary, under State regulation, since the people have some rights both in the preservation of their heritage and in maintaining the purity of the air they breathe.

Next to imperfect combustion the chief waste of coal arises in mining. The early colliers saw no value in coal in the ground, any more than early millers saw value in the flow of the stream; to them coal acquired value only through the labour in mining it, just as to the miller the stream acquired value only as head was produced by the labour of building a dam and mill. So the coal taken out in the British and German collieries was a sort of treasure trove; that left in the ground was nobody's loss. Likewise in early American mining, the coal mined merely yielded a return for labour, and the leaving of pillars, slack, and poor coal, after mining, began before coal lands were thought to have any other value than as wood lands or farm lands. Thus the incredibly wasteful methods were natural enough: if labour could be saved and profit gained by taking out but a third or a half of the richest part of the seam, leaving the rest to be rendered inaccessible by caving, so be it. No one thought of it as improvident. Now that the coal in the ground is recognised as part, and a great part, of the value of coal lands, self-interest impels the operator to take out all he can and leads the miner to work close to floor and roof.

The most promising check on coal consumption is the substitution of other power. There is abundance of power lying around us if we only knew how to harness it. It is only within the past decade that electrical transmission has made water-power generally available for driving machinery, for smelting, and for moving trains, and has at the same time created a new market for copper. And just as the woods and the ores and the mineral fuels have become sources of wealth and power within our memory, so will the running waters become within a few years." —ANDREW CARNEGIE. — *Iron and Coal Trades Review*, Nov. 6, 1908, p. 1999. (A. R.)

EFFECT OF PRESSURE ON THE BOILING POINT OF SULPHUR.—"Within the past few years it has been recognised that the platinum-resistance thermometer furnishes probably the most exact means we possess for temperature measurements over a very wide range. As is well known, the three fundamental fixed points, on which the scale of this instrument is based, are usually 0°, 100°, and the boiling point of sulphur. The latter point is used in the determination of the characteristic constant of the particular kind of wire employed in the instrument. This constant is usually called the δ of the thermometer.* The determination of this δ is made by taking the resistance of the thermometer at the temperature of sulphur vapour, boiling freely under the prevailing atmospheric pressure. Although the value of this temperature has been determined with accuracy by several different investigators for the normal pressure, its change with pressure has been always a matter of considerable uncertainty. The authors have, therefore, made a redetermination of the effect of pressure on the boiling point, carrying the range of the experiments well outside the usual atmospheric limits. Details will be given elsewhere in a later communication. The results of the experiments were found to be closely represented by the formula:

$$T = T_s + \cdot 0904\,(p - 760) - \cdot 0000519\,(p - 760)^2,$$

where T is the temperature of the vapour on the air-scale at the pressure *pmm*, and T_s the 'normal' boiling point.

The following table gives the value of the departure from normal boiling point at various pressures to the nearest ·01° C.:—

Pressure.	$T - T_s$ ° C.	Difference for 1 mm.
700	−5·61	·096
710	−4·65	·095
720	−3·70	·094
730	−2·76	·093
740	−1·83	·092
750	−0·91	·091
760	- 0·00	·090
770	+0·90	·089
780	+1·79	·088
790	+2·66	·087
800	+3·53	

It will be seen that the effect decreases somewhat rapidly with rise of pressure, and is much greater than the value usually employed, viz., ·082 mm. per degree, deduced from Regnault's observations. If, for example, the value of δ be calculated for a thermometer whose true $\delta = 1{\cdot}500$ from observations made at 730 mm., the temperature of the vapour at this pressure being obtained by use of Regnault's values for the divergence from the normal temperature, the error in the δ thus obtained would be ·018. This would lead to the scale of the instrument in use being in error by reading ·36° C. high at 500° C. and 1·6° C. at 1,000° C., quantities of importance for work of even moderate precision."—Dr. J. A. HARKER and F. P. SEXTON.—*Electrical Engineer*, Oct. 9, 1908, p. 527. (H. A. W.)

---

* This is defined by the relation

$$d = T - pt = \delta\left[\left(\frac{T}{100}\right)^2 - \frac{T}{100}\right]$$

where *d* is the difference between the 'platinum temperature,' *pt*, and the 'air temperature,' T.

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

TECHNICAL CHEMISTS' HANDBOOK. By G. LUNGE. Leather, crown 8vo, xv. and pp. 260. Tables and Methods of Analysis for Manufacturers of Inorganic Chemical Products. Price, 10s. 6d. (London: Gurney & Jackson.)

"This useful little work, which may be regarded as taking the place of Dr. Lunge's 'Alkali Maker's Handbook,' may appropriately be considered in conjunction with the 'Technical Methods of Chemical Analysis' reviewed below.

It comprises a most valuable series of tables of specific gravities, analytical factors, atomic and molecular weights, logarithms, money tables, weights and measures, etc., etc., with brief outlines of the technical methods in vogue for special analyses, particularly as regards alkali and acid manufacture.

Compressed as it is to little over 250 pages, it is avowedly intended only for the man who is fairly *au fait* with his subject, but as a 'handbook,' and particularly for its 150 pages or so of tables, it cannot be too strongly recommended."—*The London Mining Journal*, Nov. 21, 1908, p. 650. (W. A. C.)

TECHNICAL METHODS OF CHEMICAL ANALYSIS. Edited by GEORGE LUNGE, Ph.D. English Translation from the latest German edition, adapted to English Conditions of Manufacture, edited by CHARLES ALEXANDER KEANE, D.Sc., Ph.D. Vol. I., in two parts. Three vols., pp. 1,024. Medium 8vo. Price, £2 12s. 6d. (London: Gurney & Jackson.)

"This valuable work, of which the first volume out of the three which will ultimately appear, has just been issued in two parts, forms the first instalment of one of the most complete treatises on general technical chemical analysis which has yet been published. Edited in the original German edition by Dr. Lunge with distinguished collaborators, and now published as an English edition under the editorship of Dr. Keane, also with distinguished assistants, the volume before us possesses a value far beyond that of mere translation. The English edition has been altered where necessary, to meet the differences between the industrial processes, legislation, etc., of the two countries, and a perusal of its pages, coupled with the fact that those who are responsible for it are experts in the special branches which they have undertaken to look over or revise, shows that the end aimed at has been fully achieved.

Without knowing what the remaining two volumes will contain, the two parts under review may, no doubt, be taken as typical, and among the many parts which interest the mining man or metallurgist, as well as the chemist, we have important sections dealing with the examination of fuels, furnace, and other gases, glass, clay, cements, and the materials employed in the manufacture, the minerals and ores used in the various branches of chemical industry, and in many metallurgical operations such as the manufacture of aluminium, the analyses of boiler and other waters, and even of air, often an important subject of investigation for the mining man. Not only are the sections valuable to the analyst, they possess a special interest—rare in technical work—through the introduction of sufficient historical or descriptive matter to provide a wholesome leaven to the nourishing but otherwise less palatable mass of details which the reader requires to digest.

The special methods of analysis necessary for ores, furnace products, etc., and other matters relating mainly to metallurgical work are naturally but lightly treated, but, so far as they are dealt with, the point of view from which they are considered is sufficiently different from that of the usual assayer to add to rather than detract from their value as supplementary to his ordinary works of reference. The portion dealing with the analysis of cyanides gives, for instance, a resumé of the method of manufacture and many interesting details of interest even to those who only regard cyanide as a solvent for gold or silver.

Written avowedly for the technical chemist before all others, and edited and checked by those who are mainly technical chemists, one can scarcely expect the mine manager and assayer to have been specially catered for, but it may not be amiss to point out one or two points where slight alterations might be made. The statement that commercial sodium cyanide tests as a rule 125%, calculated as potassium cyanide, and sometimes 'as high as 128%,' might, for instance, be altered to 'commonly assays between 128 and 130%,' seeing that the 129% is quite a usual assay for material purchased by the ton for gold extraction.

The remark that 'native manganese ore consists of manganese dioxide,' although made under the 'chlorine industry,' and therefore correct as regards the ore used for that special purpose, seems somewhat sweeping when one remembers the hundreds of thousands of tons of manganese ore use annually in metallurgical work and consisting mainly of oxides other than the dioxide. For the same reason—viz., that the ores referred to here and in other sections are commonly only considered from the point of view of the specialist who requires them for a specific purpose, the methods of analysis given are often only those adapted for such specific use.

The above criticism would be quite unjustifiable were it not for the fact that the work forms so valuable an adjunct to the assayer's ordinary reference library as to render it advisable to add an occasional note that certain references to the ores used and their methods of analysis are entirely confined to the special uses included in the sections in which they occur.

The volumes carry a large number of valuable tables, and a useful innovation is the collection, as an appendix to the second volume, of all those which have appeared through the letterpress."—*The London Mining Journal*, Nov. 21, 1908, p. 649. (W. A. C.)

A GUIDE TO TECHNICAL WRITING. By T. A. RICKARD; pp. 127. Price, 4s. 6d. nett. (San Francisco, *Mining and Scientific Press.*)

Mr. Rickard's practical treatise on "Technical Writing" should receive a ready welcome from all members of the mining world, who have at any time the occasion to express their ideas for the benefit of others. The mining, mechanical or metallurgical engineer, who hopes to keep abreast of modern progress, is forced to study the published records of work done outside the range of his personal observation, and he should be encouraged to give in return the results of his own experiences. A work, such as the one under review, which specialises in the fundamentals of English composition as it directly concerns the technical man and indicates how lucidity of expression may be best attained and ambiguities avoided, will provide a valuable stimulant to the commerce in ideas carried on so profitably between workers in all parts of the English-speaking world of industry.

Mr. Rickard's qualifications as an authority upon technical writing need no explanation, for he is already sufficiently well known in all mining countries as a brilliant writer, and as a scientist of high attainments. What he preaches in his latest book he has long practised with literary results constituting the highest tribute to the soundness of his precepts. His object is clearly to provoke greater attention to the essentials of good technical writing rather than to lay down dogmatical rules for unthinking acceptance. Where doubt should not exist, he is, however, decisive in his opinions.

The book is well classified into twenty parts. After an introductory chapter pointing out the importance of a study of the subject, he gives some sound advice upon the avoidance of mongrel words, "spurious coin," of local application, such as "gallows' frame," "sulphurets," "ledge" and even our time-honoured "reef"; also upon the employment of abbreviations and numbers, and upon the use and abuse of hyphens, so often productive of misleading terms. A particularly interesting chapter deals with "Some Words and their Ways." As an example of common error, he mentions the use of "tank" and "vat" as synonymous, although the former should be confined to the vessel used for holding water, etc., and the latter to those used for treatment processes. "Ledge" and "reef" are localisms, Mr. Rickard contends, for which there is "no need to-day." Perhaps, however, the Rand, with its yearly output equivalent to 30,000,000 sovereigns, may be allowed to give this "spurious coin" the mark of genuineness and reserve the word "vein" for outside usage. Such a plea for localism is, strictly, unjustifiable, but as our gold deposit will never be known by any other term, and its life of productiveness is assured for many years to come, it may be advisable to sanction the word's elevation, with its bastard associate, "banket," to the rank of verbal legitimacy.

To pick out from a mass of sound restrictions, too often neglected, one or two other terms to which his anthipathy may appear slightly ungenerous, reference may be made to his paragraphs on the words "section" and "value." He objects strongly, to the employment of the former when "part or portion" is meant as in "that section of the mining camp." Section means the "view of something along an intersecting plane." Rand usage admittedly necessitates a consideration of context to show the meaning applied, but it is hard to suggest a good substitute, which fixes our thoughts to well-defined divisions of the main reef series alone to parts "cut off" from each other by great faults or dykes, or arbitrary boundary lines. The author strongly condemns the use of the word "value" or "values," for the valuable metals or minerals. We should not say that "the values improve" below the dyke, or that "80% of the values" are recovered. Yet so serviceable has this abstract been that one would be loth to see it ignominously banished from technical writings purporting to possess no enduring importance.

Being himself so forcible a writer, he leaves the reader in no doubt as to which terms and expressions are essentially wrong, and which are merely undesirable and capable of improvement.

Mr. Rickard closes his little book with some parting advice in the shape of six golden rules. The last and most important is, simply, "remember the reader." Elsewhere he urges all writers to "sacrifice elegance, even modesty, to directness of statement." (R. S.)

THE CHEMICAL ANALYSIS OF IRON. By ANDREW A. BLAIR. Seventh Edition. Pp. 334, illustrated. 6½ × 9 in. : half leather, $4. (Philadelphia and London, 1908 : J. B. Lippincott Company.)

"The former editions of this work and the standing of the author as an analyst are too well known to require any comment. The seventh edition of this valuable contribution to the analysis of iron, steel and blast-furnace materials differs from previous editions by the addition of methods for the separation of vanadium, molybdenum, chromium and nickel in steel ; an account of the volumetric method for nickel, and some changes in the methods for gas analysis. The table of factors has been recalculated from the values for 1908, as given by the International Committee and the corrected factors have been substituted throughout the text."—*Engineering and Mining Journal*, Dec. 5, 1908, p. 1115. (W. A. C.)

The Science Year Book, 1909. With Astronomical, Physical and Chemical Tables, Summary of Progress in Science, Directory, Biographies, and Diary for 1909 ; 5th year of issue. Edit. by Major B. F. S. Baden-Powell. Frontispiece. Demy 8vo., pp. 365. King, Sell & Olding. Net 5s.

Cockin, T. H. Elementary Class-Book of Practical Coal Mining : 3rd edition. Revised. Cr. 8vo., pp. 440. Lockwood. Net 4s. 6d.

Gale, H. Geology of the Rangely Oil District, Rio Blanco County, Colorado. With a section on the Water Supply. (Bull. U.S. Geol. Survey.) 2 plates and 2 maps. W. Wesley. Net 2s.

Griswold, W. Structure of the Berea Oil Sand in the Flushing Quadrangle, Harrison, Belmont and Guernsey Counties, Ohio (Bull. U.S. Geol. Survey). 2 maps. Roy. 8vo. sd., pp. 30. W. Wesley. Net 1s.

Hayes, C., and Lindgren, W. Contributions to Economic Geology, 1907. Part I., Metals and Nonmetals, except Fuels. (Bull. U.S. Geol. Survey.) 5 plates and engravings. Roy. 8vo. sd., pp. 482. W. Wesley. Net 4s.

Leith, C., and Harder, E. The Iron Ores of the Iron Springs Districts, Southern Utah. (Bull. U.S. Geol. Survey.) 20 plates and 5 maps. Roy. 8vo., sd. W. Wesley. Net 2s. 6d.

Logan, J. A. Practical Mining in South Africa. Cr. 8vo., pp. 252. J. Jones, Durban. Net 10s. 6d.

Modern Mining Practice. Edit. by George Mitcheson Bailies. Vol. 4. Roy. 8vo., pp. 272. J. H. Bennett.

O'Donahue, T. A. Colliery Surveying. A Primer designed for the use of Students and Colliery Manager Aspirants. New ed. Cr. 8vo., pp. 268. Macmillan. 3s. 6d.

Prindle, L. The Fairbanks and Rampart Quadrangles, Yukon-Tanana Region, Alaska. With a Section of Rampart Placers by F. Hess and a paper on the Water Supply of the Fairbanks Region by C. Covert (Bull. U.S. Geol. Survey). 3 plates and 2 maps. Roy. 8vo., sd., pp. 102. W. Wesley. Net 2s. 6d.

Ransome, F., and Calkins, F. The Geology and Ore Deposits of the Coeur d'Alene District, Idaho (Professional Paper, U.L. Geol. Survey). 29 plates, 2 maps and engravings. 4to., sd., pp. 203. W. Wesley. Net 8s.

Jamieson, Andrew. A Text-book on Steam and Steam Engines. 16th ed. Revised. Cr. 8vo., pp. 852. C. Griffin. 10s. 6d.

Liversidge, John G. Engine Room, Practice. 6th ed. Revised. Cr. 8vo., pp. 404. C. Griffin. Net 6s.

Wimperis, H. G. The Internal Combustion Engine. Being a Text-book on Gas, Oil and Petrol Engines. For the Use of Students and Beginners. 8vo., pp. 340. Constable. Net. 6s.

Kerr, George I. Elementary Coal Mining. 2nd ed. Revised. Cr. 8vo., pp. 234. C. Griffin 3s. 6d.

Scientists' Reference Book and Diary for 1909. 18mo. Woolley (Manchester). 1s.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 499/08. J. Bowyer. Improvements in means for diverting the pulp discharge from a mortar box on to the amalgamated plate of an adjacent box. 19.12.08.

(P.) 500/08. A. J. Arbuckle. Improvements in apparatus for the treatment of crushed ore products for the recovery of the metal contents. 19.12.08.

(P.) 503/08. Rollo Bowman Ballantine. An improved process for the manufacture of cyanides. 21.12.08.

(P.) 504/08. Walter Hooper Jane. Thrust collar for cam shafts and the like. 22.12.08.

(P.) 506/08. James Chalmers Johnstone. Improvements in guide blocks for the stamps of stamp mills. 23.12.08.

(C.) 507/08. Albert Groth (1), Herbert Fuller Carter (2). Improved process for filtering and washing the filtrate and apparatus therefor. 23.12.08.

(P.) 509/08. Samuel James Davis. Device for stopping runaway skips and the like. 23.12.08.

(P.) 512/08. Charles Robert Bishop. A new and improved lubricant mixture. 23.12.08.

(P.) 514/08. Paul Kaehn. An improved rocker tipping device for trucks of various styles. 28.12.08.

(C.) 516/08. Albert De Dion (1), George Bouton (2). Improvements in or relating to pressure-bearings for bevel gears and the like. 28.12.08.

(C.) 517/08. Albert De Dion (1), George Bouton (2). Improvements in or relating to fans for motor vehicles. 28.12.08.

(C.) 518/08. John Hutchings. Improvements in and relating to direct acting pumping engines. 28.12.08.

(C.) 519/08. Francis Edward Elmore (1), Alexander Stanley Elmore (2). Improvements in apparatus for more or less completely separating subdivided solid matter from a liquid, particularly applicable for thickening ore pulp and for feeding or measuring the pulp or solid. 28.12.08.

(C.) 520/08. The Hon. Charles Algernon Parsons. Improvements relating to turbines. 28.12.08.

(C.) 521/08. Edward Thomson. Improvements in belting. 28.12.08.

(P.) 522/08. Hector William Cole (1), Robert Ernest Ireland (2). Safety gear for skips, cages, elevators and the like. 29.12.08.

(P.) 523/08. Edwin Frank Kruspe. Improvements in means for driving the ore feeders of stamp mills. 29.12.08.

(C.) 525/08. Henry Edward Dilke Merry. An improvement relating to the feed screw of rock drills. 29.12.08.

(P.) 527/08. Raphael Michal-ange Chevallier. Improvements in tappets for actuating the valves of engines or motors. 31.12.08.

(C.) 1/09. Walter Skelton Hudson (1), Edward Hudson (2). Improvements in or relating to roller bearings. 2.1.09.

(P.) 2/09. William Badger. A compound alternate double impeller, centrifugal, eclipse, sand, slimes and tailings water pump. 4.1.09.

(P.) 3/09. Edward Thomson. Improvements in dies and mullers for grinding and amalgamating gold and silver ores. 5.1.09.

(P.) 4/09. D. Frame. Wedgeless safety hammer and hand tool shafting. 6.1.09.

(P.) 5/09. Thomas Feaster Morgan. An improved guide roller or carrier for conveyor belts, transmission belts and ropes. 7.1.09.

(C.) 6/09. Walter Scheurmann. Improvements in balancing device for centrifugal pumps and the like. 9.1.09.

(C.) 7/09. Walter Scheurmann. Improvements in air and gas pumps and compressors and the like. 8.1.09.

(P.) 8/09. Harry August Wilson. A new or improved combined automatic depth and weight indicator for use on mines. 8.1.09.

(C.) 9/09. Colman Arthur Terry. Improvements in and relating to core drilling apparatus. 8.1.09.

(C.) 11/09. David Honeywood (1), Farratt Payne (2). Improvements in machinery for crushing or grinding ore, rock, cement, and the like. 8.1.09.

(C.) 12/09. Max Kurth. Improvements in variable volume fluid compressors. 8.1.09.

(P.) 13/09. Albert Benz. Improved machine for rolling rod iron. 8.1.09.

(P.) 14/09. Alexander John Arbuckle. A new or improved protective cover for preventing unauthorised access to the surfaces of amalgamating plates, the interiors of extractor boxes and the like. 9.1.09.

(C.) 17/09. Charles Christiansen. Improvements in rock drilling machine valves. 11.1.09.

(P.) 20/09. Norman Mathieson. Improvements in rope jockeys. 13.1.09.

(P.) 21/09. Frank Gowine Alfred Roberts (1), James Frederick Ferguson (2), Alwin Rupert Troye (3). Improvements in assay furnaces. 13.1.09.

(P.) 22/09. William Taylor. Concrete lining for tube mills. 13.1.09.

(C.) 23/09. Frank Cotton. An improved method of and means for burning fuel for firing steam boilers. 14.1.09.

(C.) 24/09. Charles Christiansen. Improvements relating to hammer rock drills. 15.1.09.

(C.) 26/09. Alfred Ernest Terry. Improvements in screw locking nuts. 14.1.09.

(P.) 27/09. Frederick Hudson Shepherd Shepherd (1), Christiaan Stephanus Hugo (2). Improvements in rock drilling apparatus or machines. 16.1.09.

(P.) 28/09. Hans Charles Behr. Improvements in stamp lifting means. 16.1.09.

(P.) 29/09. Forest Gale Parsons. Improvements in means for automatically controlling hauling, winding and like machines. 19.1.09.

(P.) 30/09. William Taylor. An improved method of extracting gold from quartz and ores containing same by means of bituminous substances. 20.1.09.

(C.) 31/09. Henry Hay (1), Bruce Edward Tennent (2). A new or improved material for precipitating gold from chlorine or cyanide solutions and method of manufacturing the same. 20.1.09.

(C.) 32/09). Henry Hay (1). Bruce Edward Tennent (2). Improvements in the precipitation of gold from chlorine and cyanide solutions. 20.1.09.

(C.) 33/09. William White. Improvements in systems of washing and filling locomotive and other boilers. 21.1.09.

(P.) 35/09. Fabian Mercer Cox. An improved air-lock caisson and appliances for the recovery of diamonds, precious stones and metals from deep waters. 22.1.09.

(C.) 36/09. Alfred Arthur Lockwood (1), Marcus Reginald Anthony Samuel (2). Improvements relating to the treatment of ores. 22.1.09.

(C.) 37/09. Alfred Arthur Lockwood (1), Marcus Reginald Anthony Samuel (2). Improvements relating to the treatment of ores or the like. 22.1.09.

(C.) 38/09. Dr. Robert Gans. Process for the extraction of gold from auriferous solutions or liquids. 22.1.09.

(C.) 39/09. Ernst Schmatolla. Improvements in kilns for burning limestone, dolomite and the like. 22.1.09.

(P.) 40/09. Henry Haugh. Improvements in filter presses. 23.1.09.

(P.) 41/09. James Miners Holman (1), John Leonard Holman (2). Improvements in means for actuating shaking conveyors. 23.1.09.

(P.) 42/09. Norman Mathieson. Improvements in lining means for tube mills. 25.1.09.

(P.) 43/09. Frederick Dwight Johnson (1), Henry Samuel Potter (2). Improvements in and relating to rock drills. 25.1.09.

(P.) 44/09. Charles Reeves (1), Thomas Edwards (2). Cantilever platform for tipping tailings, ashes, earth, refuse, etc. 28.1.09.

(P.) 45/09. George Tom Cannon, M.R.C.V.S. Improvements in jockey or rope grips for mechanical haulage systems. 28.1.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

BRADFORD, WAGER, *to* Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg.
COLVILLE, C. A. B., *l/o* Bulawayo; P. O. Box 158, Salisbury.
COOMBE, M. H., *to* P. O. Box 1145, Johannesburg.
HALDANE, Dr. J. S., *to* Cherwell, Oxford.
HAUGH, H., *l/o* Rhodesia; c/o Messrs. Towns & Co., Loftus Street, Sydney, N.S.W.
TAYLOR, W., *l/o* Johannesburg: Malvern, P. O. Denver.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. IX. FEBRUARY, 1909. No. 8.

Proceedings
AT
Ordinary General Meeting, February 20, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, February 20th, Mr. R. G. Bevington (President), in the chair. There were also present:—

37 Members: Dr. J. Moir, Messrs. A. McA. Johnston, E. H. Croghan, K. L. Graham, A. Richardson, G. O. Smart, Prof. G. H. Stanley, A. Whitby, H. A. White, Prof. J. A. Wilkinson, D. J. Arkell, W. Beaver, H. D. Bell, E. Blume, M. H. Coombe, G. A. Darling, A. D. Gilmour, J. C. Greer, J. H. Hamill, Dr. J. McC. Henderson, E. Hollis, A. G. Hoyer, Hy. Lea, J. Lea, J. McLennan, J. Pyles, S. C. Quin, W. H. Roe, O. D. Ross, H. A. Scarf, S. Shlom, R. Stokes, A. Thomas, O. Tonnesen, J. F. Walker, and Dr. E. Weiskopf.

10 Associates and Student: Messrs. A. R. Adams, J. Chilton, H. G. Kirkland, A. Locke, C. L. Moon, R. Murdoch, D. Nicholas, W. A. C. Tayler, W. Waters, and J. Whitehouse.

12 Visitors, including Dr. Mehliss, Messrs. D. Greig and J. A. Vaughan; and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the January *Journal*, were confirmed.

NEW MEMBERS.

Messrs. J. Whitehouse and W. Beaver were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows:—

Allen, Robert, M.A., B.Sc., M.I.M.M., Villa Escobedo, Chihuahua, Mexico. Metallurgical Engineer.
Barclay, George Moore, Knights Deep, Ltd., P. O. Box 143, Germiston. Mining Engineer.
Bowers, H. O., Robinson G. M. Co., Ltd., P. O. Box 1024, Johannesburg. Mining Engineer.
Busch, Gustave Edgar, New Lisbon-Berlyn, Ltd., Pilgrims' Rest. Mining Engineer.
Cantor, Bertie Charles Moxon, Blanket Mine, Gwanda, Rhodesia. Mining Engineer.
Cropper, Cecil Howe, c/o Institution of Mining and Metallurgy, Salisbury House, London, E.C. Mining Engineer.
Dodd, Herbert, City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg. Mine Captain.
Ford, F. C., P. O. Langlaagte. Cyanider.
Healey, John Edward, East Rand Proprietary Mines, Ltd., P. O. Box 145, East Rand. Mining Engineer.
Neal, Walter, Las Dos Estrellas, Apartado 34, El Oro, Mexico. Superintendent Cyanide Department. (*Transfer from Associate Roll.*)
Philp, Henry, Lebong Donok, Benkoelen, Sumatra, Dutch East Indies. Underground Manager.
Robson, John Newton, Nigel G. M. Co., Ltd., P. O. Box 20, Nigel. Mill Foreman.

The Secretary: Since the last meeting of the Society the following associates have been admitted:—

Dick, Otto Emile, Geldenhuis Deep, Ltd., Cleveland. Battery Foreman.
Townsend, Charles Thomas, Brilliant Extended Cyanide Works, Charters Towers, Queensland. Metallurgical and Cyanide Manager.

General Business.

The Secretary: A number of invitations were sent by the Council to leading gentlemen here, and to the Executive of the Chamber of Mines and others to attend this meeting, but unfortunately, owing to the annual dinner of the Mine Managers' Association, they have been unable to attend. I have received letters of regret from Messrs. Lionel Phillips, J. G. Hamilton, M.L.A., H. J. Hofmeyr, R. N. Kotzé, G. W. Higgins, C. Chudleigh, L. Reyersbach, W. H. Dawe, E. Goffe, W. Adye and J. Cowie.

The President: I should like to welcome to our meeting Mr. W. F. Lance, M.L.A., the Chairman of the Council of the University College. This is our first general meeting in the building of the University College, and in this lecture theatre. At our last monthly meeting, I drew your attention to the probability that our meetings would be very much more interesting and instructive when held here, because there would be the possibilty of demonstrating papers read before the Society, which has hitherto been impossible. As you see, there is a table which is provided with appliances for demonstrating chemical or metallurgical processes which may be brought before you, and I am sure you will appreciate the fact that we are now meeting in an institution giving us such facilities.

Mr. W. F. Lance, M.L.A., responding, said: Mr. President and gentlemen, I thank you for your welcome. I may say I am very glad to avail myself of your kind invitation to be present to-night, because it affords me an opportunity of offering you, on behalf of the Council of the University College, a very cordial welcome to these premises, which we wish you to regard as your home. It seems to me a very fitting thing that your Society, closely associated as it is with that great mining industry on which this town lives, should be the first of the scientific societies to hold their meeting in this building, a building which is a gift to the town of Johannesburg, through the University College, of the Witwatersrand Council of Education, and the late Crown Colony Government. The Council of the University College, I need hardly tell you, holds this building for educational purposes, amongst which is specificially included the housing of the various scientific societies, and therefore we are glad to welcome you here to-night. The trust is one which I feel sure the Council of the College will do its utmost, and be only too pleased, to loyally carry out. Gentlemen, it is not for me to detain you to-night. You have, what appears to me as a layman, a somewhat terrible programme to get through, therefore, I will only repeat, that you are to-night as a Society in your own home, and I hope that the sense of security and comfort arising therefrom may long avail you and tend to increase your sphere of usefulness.

MINES TRIALS COMMITTEE.

The Secretary read the following letter:—

P. O. Box 1167,
Johannesburg.
January 23, 1909.

F. Rowland, Esq.,
Secretary, Chemical, Metallurgical
and Mining Society,
P. O. Box 1183, Johannesburg.

Dear Sir,

With reference to my previous letters, I have been instructed to inform you that the scope of my Committee's operations has now been extended in order to include the consideration of mining and mechanical improvements.

My Committee would be obliged if you would kindly make this known to the members of your Society.

Thanking you in anticipation.

Yours faithfully,
(Sgd.) John P. Long,
Secretary, Mines Trials Committee.

The President: You will notice that the scope of the Mines Trials Committee has now been extended. I made rather a *faux pas* in my inaugural address before the Society. I stated that the Mines Trials Committee dealt with the winning of rock and ore from the very commencement. I was somewhat pulled over the coals for that, and was told that the Mines Trials Committee only dealt with the winning of gold from the collar of the shaft. I then said I was very sorry. Now the Mines Trials Committee seem to have been sorry also, and I am glad they have extended their scope.

MINE DISASTERS.

The President: There is a matter to which I wish to allude; one which must arouse in us, as members of a mining society, feelings of regret and sympathy; that is the large number of mining disasters which have quite recently been happening in various parts of the world, unfortunately frequently attended with serious loss of life. Our own country, the Transvaal, has not escaped, but has, I think, had its full quota of trouble in this respect. In dealing with the situations resulting from these accidents a great deal of arduous and plucky work has been carried out by some of our members and confrères in their efforts to save life, attended with, one is glad to know, a large measure of success. While all rescue work is looked upon by mining men as but a part of their common duty, mankind in general cannot but admire the pluck in facing danger which has been shown, and without individualising in any case, I feel sure the members of our Society will agree with me that it is but right that we should place on record our appreciation of the good services which have been rendered in the cause of humanity; as well as with sadder feeling, tendering our sincere sympathy to the relatives and friends of those who have unfortunately lost their lives.

INSTITUTION OF MINING AND METALLURGY.

The Secretary: The President has received the following letter from Mr. Alfred James, President of the Institution of Mining and Metallurgy.

Salisbury House,
London, E.C.,
January 13, 1909.

The President,
The Chemical, Metallurgical and
Mining Society of S A.,
Johannesburg, Transvaal.

Sir,—The friendly co-operation of the sister societies in Australia, Canada and South Africa, with the Institution of Mining and Metallurgy in its efforts for the advancement of the best interests of the mining and metallurgical industries has, in the opinion of the Council, been productive of lasting benefit to the profession and the industries.

Recognising this fact, and in the hope of still further cementing the friendly relationships, happily subsisting between the Institution and the three principal Colonial societies, the Council have unanimously resolved to invite the Presidents for the time being of each of the following societies to accept, *ex officio*, election as Honorary Members and Corresponding Members of Council of the Institution during their respective terms of office :—

The Australasian Institute of Mining Engineers.
The Canadian Mining Institute.
The Chemical, Metallurgical and Mining Society of South Africa.

The Council trusts that you may be able to accept this invitation on your own behalf, and on that of your successors in the Presidency of the Chemical, Metallurgical and Mining Society of South Africa.

I enclose for your information a copy of the Bye-Laws and List of Members of the Institution, on p. 23 of which you will find the names of the present Honorary Members of the Institution, and I append to this letter a short memorandum on the functions of Corresponding Members of Council for the appointment of whom Bye-Law 1a, Section V., was adopted at a Special General Meeting of the Institution, held in March last.

I have the honour to be,
Sir,
Your obedient Servant,
(Sgd.) Alfred James,
President, The Institution of
Mining and Metallurgy.

THE INSTITUTION OF MINING AND METALLURGY.

Corresponding Members of Council.

Excerpt from Minutes of Council dated Nov. 18, 1908.

"That Corresponding Members of Council shall, when in Great Britain, receive notices of Council Meetings, and shall have the privilege of attending and voting at such meetings."

"That the corresponding members of Council in any locality shall act as a Commitee of Council for their district, and shall have power to appoint local Consultative Committees of ordinary Members or Associates of the Institution, to advise on matters referred to them by the Council, or on matters of local interest which the committees may themselves wish to submit for consideration by the Council."

The President : You may be sure that I have replied to this letter, accepting with gratitude and with a sense of the honour done to myself, the position of Corresponding Member of the Council of the Institution of Mining and Metallurgy, of which I have the honour to be a member. Also on behalf of the future Presidents of this Society, I have assured the Institution of Mining and Metallurgy, that they will feel the great honour which has been done them and the Society.

TECHNICAL SOCIETIES COMMON ROOM.

There is one further matter which has been alluded to two or three times in our meetings, that is the establishment of a common room in this building as a common meeting place for the members of the various scientific societies. That is now an accomplished fact, and by the end of this month a room will be ready for your occupation, and I feel sure it will be of great use to members of technical societies. If at any time they have a few moments to spare, they can come in here. The room is on the ground floor, where they will find comfortable chairs, periodicals and papers of various descriptions, and where they can meet the members of other societies, and thus further that co-operation which was spoken of at our last annual meeting. I think, on behalf of the Society, we ought to tender our thanks to the Council of the Transvaal University College for having set aside a room for this purpose, and I hope that the members of this Society in particular will make good use of it, and thereby show their appreciation of the course adopted by the Council of the Transvaal University College.

ILLNESS OF COUNCIL MEMBERS.

The President : You will remember that our Hon. Treasurer, Mr. Littlejohn, has been very ill, and has undergone a very serious operation, and it is somewhat of a wonder that he has pulled through. Our Secretary has just telephoned to the hospital and learns that Mr. Littlejohn is progressing, but progressing very slowly. I am sure that we must all hope that his progress, though slow, may be sure, and that within a reasonable time he may be restored to our Society and in the full enjoyment of health. You will also be aware of the serious accident which has befallen another member of our Council, Mr. C. B. Kingston, in having his thigh broken. Another white man was similarly injured, who afterwards died, and two or three natives were killed alongside of him. We hear by telephone that Mr. Kingston is progressing very well indeed. I am sure that we shall all sympathise with him in the serious accident he has sustained, and I have no hesitation in saying that we are all pleased to hear that he is going on so well.

I will now ask Prof. Stanley to read his note on "A Cheap Platinum Parting Apparatus."

A CHEAP PLATINUM PARTING APPARATUS.

Prof. G. H. Stanley (*Member of Council*): In bullion assaying on these fields the use of platinum parting apparatus appears to be the exception rather than the rule, in spite of the advantages which accrue. The chief reason for this is undoubtedly the heavy initial cost of the ordinary forms, in view of the comparatively small number of assays usually necessary per month, on any individual mine. Some years ago I encountered the same difficulty, and, I think, surmounted it, and therefore venture to place a description of the apparatus before the Society in the hope that it may prove useful. It can be made by any assayer of ordinary deftness, for himself, at a cost of a few shillings, and with care, will last well; the thimbles exhibited I made over five years ago, though I need hardly say they have not been in constant use *

The term "platinum" parting apparatus is perhaps rather misleading, inasmuch as only a comparatively small part of the outfit, namely, the thimbles and wire supports, are made of platinum. These latter are formed as follows: A piece of platinum foil, about 2 in. × 1 in. has its shorter edges doubled over for about $\frac{1}{8}$ in. to obtain greater strength, and is then placed with its centre over the rounded end of a pattern such as a small test tube or glass rod, $\frac{1}{2}$ in. in diameter, then carefully bent over as shown in Fig. I. (i. and ii.), the edges being lapped over one another, and creases smoothed out by some smooth hard body. A slit (or +) is cut on the bottom while on the pattern, and a small hole, *b*, made on each side in such a position that the wire afterwards attached as support, will secure the two portions together. The thimble thus formed is removed from the pattern and the ends of a piece of platinum wire, about 2 in. long, pushed through these holes and carefully twisted over tight to form the suspending loop, this resulting in the production of the thimble shown in Fig. I. (iii.) The thimbles are in turn suspended from supports made by bending glass rod to the form shown in Fig. I. (iv.) and fusing small loops made of platinum wire into the ends, the span at the bottom depending on the size of beaker, and that in turn on the number of assays per beaker—details varying in almost every case.

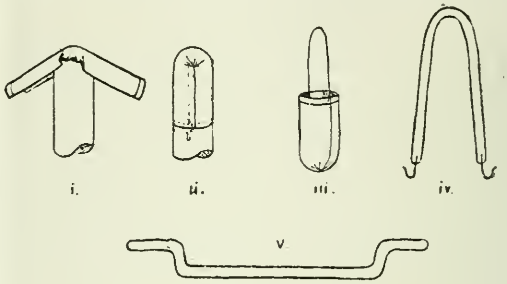


FIG. I.

These hooks again are carried on a cross bar of the shape shown Fig. I. (v.), also made of glass rod, the bends being placed conveniently to rest inside the beaker containing the acid, the depth of bend being sufficient to allow a cover glass to rest on the beaker, and the beaker being provided with a second spout, as nearly as possible opposite the first, to take an end of the carrying rod. (This second spout is easily formed by heating and bending the edge.)

The apparatus is then assembled as shown in Fig. II., a 400 c.c. Jena beaker easily taking eight thimbles. The mode of employment is almost self-evident; a beaker is charged with the required amount of acid and heated

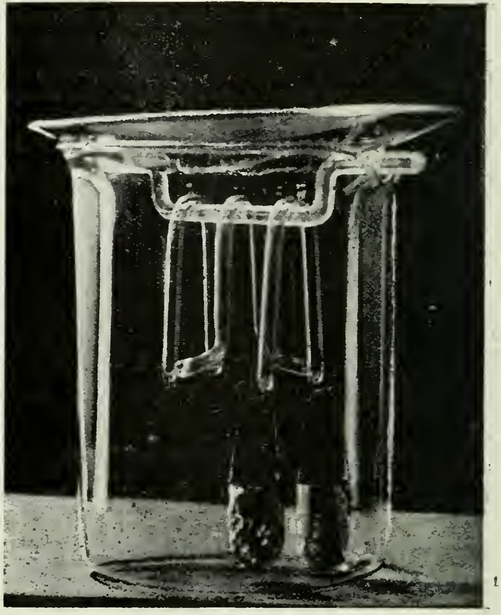


FIG. II.

---

* This platinum foil is 0·02 mm. thick.

nearly to boiling, the thimbles containing the cornets, already in position on the hooks and rod, are then introduced and the beaker covered by a glass, which may be water-cooled if desired, to lessen loss by evaporation. The temperature is then increased and boiling allowed for the required time, after which the thimbles are lifted out by means of the glass rod support, still together, rinsed by dipping into distilled water and again charged simultaneously into the second acid, already heated in a second beaker.

When the second boiling is finished, they are removed, washed by dipping into water and moving gently about for a few seconds (still together) lifted out, drained. then lifted off their supporting hooks one by one (by means of a hook of platinum wire fused into a glass rod), excess water removed by touching the thimbles down on to blotting paper, and each thimble with its cornet then placed in separate annealing cups or porcelain crucibles, which may be charged immediately into the muffle for annealing.

Annealing, as is necessary with all platinum apparatus, should be carried out at a fairly high temperature to avoid sticking, and after cooling the cornets are picked out and weighed as usual.

Various modifications will, of course, suggest themselves: the thimbles could be carried, for instance, by a glass tripod stand provided with twisted loops of platinum wire, and an improvement, though at increased cost, would be to replace the whole of the glass supports in either instance by stout platinum or gold wire.

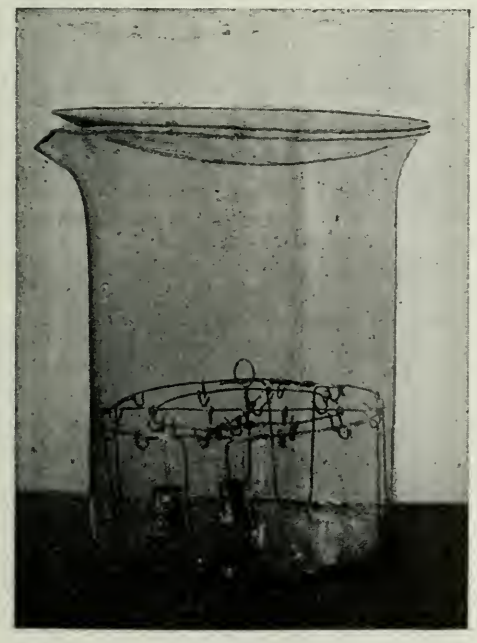


Fig. III.

On these lines the second apparatus shown in Fig. III. has been designed, and is more advantageous for dealing with larger numbers of assays, and also because the individual charging of thimbles into the muffle for annealing is avoided, since the whole supporting framework (if made of platinum) may go into the muffle carrying its thimbles. The principle of supporting the thimbles by loops from hooks is retained, but the supporting arrangement consists of a stand made of stout wire, with hooks of the same material attached to the horizontal members and so spaced that any thimble may be removed independently if desired.

In the example shown (a model in copper), the wire frame is in the form of two concentric circles, 4·5 and 3 in. in diameter respectively, attached to the straight wires forming two diameters at right angles, the whole being carried by attached wire legs 2 in. high and provided with a central loop for handling; the wire is ·035 in. thick. As shown, it takes 24 cornets and its weight (in platinum) is 36·5 grm. without thimbles. If desired, by placing the supporting hooks more closely together, about 40 thimbles could be carried by the same sized frame.

Leaving this subject, I should like to draw your attention to the examples of chemical apparatus made of fused silica which I have on the table before me, and which arrived from England, where they were manufactured, yesterday. You are, doubtless, aware of the advantages of this material, but till comparatively recently it was made by use of the oxy-hydrogen blowpipe, and the cost was almost prohibitive, certainly so for ordinary purposes Now, by using an electric furnace for fusion, the cost of production has been so considerably reduced that the price is not very much above that of porcelain, and its use will doubtless rapidly increase.

The President: We are very much indebted to Prof. Stanley for his exhibition of the apparatus. Of course, many of us have been used to handling platinum apparatus for many years, but its initial cost has prevented its general adoption. The method shown of manufacturing a thimble is certainly a very cheap and simple one, and may easily be carried into effect by assayers on many mines where they may be allowed to obtain a small amount of platinum foil, but where the authorities would not permit of getting an expensive platinum apparatus. I think it is very appropriate that a Professor of the Transvaal University College has been the first to open the ball at our first meeting in this building, and to give us a little demonstration. With regard to the silica apparatus, of course, the reduction of working costs comes in everywhere, and if these are going to be so very much cheaper than the apparatus made by the older method, they will be found very useful.

A Member: May I ask if Prof. Stanley uses any particular thickness of platinum foil for the purpose?

Prof. G. H. Stanley: Just as you get it from the makers out of stock.

Mr. A. Whitby (*Member of Council*): I should like to ask the author if he has made an attempt to weld the edges of these cups? I think if you put the cups on an iron rod, direct the flame of a blow-pipe on the overlapping edges and give a smart tap with a hammer, you should be able to weld them and thus give greater rigidity.

Prof. G. H. Stanley: I did not think it necessary.

Prof. J. A. Wilkinson (*Member of Council*): In support of Mr. Whitby's statement, I may say that I have welded platinum foil in that manner when making small cones for filtering purposes. By bending the foil round a steel cone, then heating, and whilst still hot giving it a smart tap with the hammer it can be made to adhere.

THE ESTIMATION OF CARBON MONOXIDE IN MINE GAS.

By E. H. Weiskopf, Ph.D., F.C.S. (Member).

Mr. Cullen, in his paper on "The Gases resulting from the Use of High Explosives,"* foreshadowed the reading of another, dealing with the methods by which the analytical results involved were obtained.

The work carried out may conveniently be considered under three distinct headings, viz.,

Firstly: The gas-volumetric determination of the products of explosion and of mine air.

Secondly: The estimation of carbon monoxide.

Thirdly: The calculation of the results obtained and their practical application.

From the title of my paper you will observe that I intend to deal with the estimation of carbon monoxide only, and am thus only partly redeeming the pledge given by Mr. Cullen. I consider it advisable therefore to explain my reasons for deviating from the original plan. With regard to the first sub-division of the subject, the principles and methods of the exact volumetric analysis of the gases involved have been so exhaustively dealt with in the various standard works, that I am sure you will not suspect me of having been foolish enough to strike out on my own and try to improve on the methods of such master-minds as Bunsen, Winkler or Hempel. A recapitulation of their methods, therefore, would serve no useful purpose. Although I have not followed the classical methods of Bunsen, who for exact analysis avoids even the employment of liquid absorbents, I may conscientiously say that every *finesse* and improvement which literature and experience suggests has been taken advantage of. If I say that the gases were displaced from the original sample bottles by means of mercury; that they were kept throughout the analysis in a water-saturated state; that they were collected and measured over mercury in a carefully calibrated measuring tube of 140 c.c. capacity; that this tube was placed inside a water jacket, whose temperature was read to within $\frac{1}{10}$° C.; that the levelling and reading of the mercury in the measurer was accomplished by means of a cathetometer to within 0·05 c.c. and that the barometric pressure, corrected for temperature and expansion of scale, was read to within 0·05 mm., I feel sure that you will share my conviction that a high degree of accuracy in the results obtained has been achieved. Beyond this, and apart from the fact that our results are more accurate than those obtained by previous workers on these fields, they have brought out no new or noteworthy facts, excepting perhaps our inability to find olefines in the mine gases. In consideration of all the foregoing I trust you will agree that a more detailed elaboration of the gas-volumetric methods employed would be mere waste of time.

With reference to the third sub-division of the paper, *i.e.*, the calculation of the results obtained and their application to practice, I do not propose to weary you with the details of the stoichiometric calculations, but I may state that in the appendix to this paper will be found a complete record of actual figures and data involved in the course of the analysis of a mine gas before and after blasting.

Particular attention is invited to the method of determining the quantity of gas taken for analysis in the gravimetric estimations of carbon dioxide and monoxide, which is a departure from the usual. As doubts have been expressed as to its accuracy, I have thought it advisable to elaborate it fully in the appendix. As to the stating of the analytical results in a manner suitable for applying them to practice, there is to my mind only one way, namely, the statement of gases actually evolved in the drillhole. The mere record of having found, say, 1·2% CO in the sample may be a correct expression of the analytical results, but it conveys nothing to us, because there is no apparent connection with any of the essential points at issue. These, to my mind, from the practical, the physical and the chemical points of view are :—

---

* See this *Journal*, November, 1908, p. 144.

To what extent does the use of a known kind and amount of explosive vitiate the mine air ?

To what extent are the results affected by the usual methods of charging, tamping and firing ? and

To what extent are the results affected by the nature and composition of the explosive used ?

The first of these points, involving, as it does, the health conditions underground, is, of course, of paramount interest. The natural remedy is efficient ventilation, and the whole subject might perhaps be left at that. To the scientist, however, the two other points are just as important and probably more interesting, and this for the reason that a proper understanding of their causation may lead us to do even better than cope with the noxious gases once liberated, namely *to prevent their formation.* There is yet another aspect of the question which offers a good deal of temptation to many of us, and that is, the great scope it affords for the evolution of fantastic theories. On this subject I shall have to offer a few remarks later on, but meantime I shall proceed with the main purpose of my paper, which is, to prove that carbon monoxide is formed in considerable quantities in blasting operations underground. The importance of this really requires no elaboration, but it may be as well to recall the fact that Mr. Mann, who carried out a very elaborate investigation in Western Australia on behalf of a Royal Commission on the "Ventilation and Sanitation of Mines," came to the conclusion that under normal conditions carbon monoxide is not formed, or only in inappreciable quantities. On the other hand, to Dr. Moir and Mr. Heymann, in their investigations carried out for the Miners' Phthisis Commission, belongs the credit of having been the first, in this country at any rate, to prove the presence of carbon monoxide in mine gases : and if their labours have been less appreciated and more sceptically received than they deserve, the inconsistency of the results in some cases, probably accounts for it. Generally speaking, there are at the present moment very many people who are inclined to doubt the fact that considerable amounts of carbon monoxide are formed underground, and to convince them that there is no possibility of error and that the methods and apparatus employed by us for its estimation are in principle and application correct, shall be my endeavour to-night.

Sampling, no matter for what purpose, is a most important step in any investigation, and in this case particular pains were taken to eliminate al sources of error. Although we desired to interfere as little as possible with the ordinary underground conditions, we considered it of advantage to obtain the gases given off by the explosion in as concentrated a state as possible, not only to increase the accuracy of the final results, but also to minimise the risk of overlooking some of the possible products, which might be present in small fractions of 1 per cent. only. The high percentage of CO_2 found in the samples is sufficient evidence that this was achieved. The actual method of sampling employed has been described by Mr. Cullen in his recent paper[*] and it is therefore unnecessary to add anything further. It might, however, be of interest to note, that although the sample bottles throughout were provided with glass taps there was, contrary to textbook opinion, no trouble with them. This is probably due to the very superior make which manufacturers are able to supply nowadays, and also, to the precaution of tying down the keys with rubber bands, thus ensuring against accidental loosening or moving.

After many experiments with various methods for estimating carbon monoxide, the iodine pentoxide method was finally decided upon. This method is based on the fact first observed by Ditte in 1870,[†] that the iodic acid or its anhydride, oxidises carbon monoxide to carbonic acid, liberating at the same time an equivalent amount of iodine.

La Harpe and Reverdin described a method for detecting CO in air as early as 1889 by[‡] passing it over iodic acid contained in a flask, which was heated to 150° C. The vessel was provided with a bent side tube which dipped into starch solution. The authors found that one to two parts of CO per thousand showed appreciable colouration due to liberated iodine.

That the reaction involved :—

$$5CO + I_2O_5 = I_2 + 5CO_2$$

is very sensitive and applicable for the quantitative estimation of traces of CO in gases, was first of all demonstrated by Maurice Nicloux[**] in April, 1898. He employed iodic acid at a temperature of 150° C. for the oxidation of the carbon monoxide and realised the importance of removing carbon dioxide, sulphur dioxide, and sulphuretted hydrogen from the gases before passing through the iodic-acid tube. He estimated the amount of iodine liberated by comparison with solutions containing known amounts of iodine in carbon-disulphide.

Armand Gautier[††] referring to this publication, stated that he had employed this method for the past seven or eight years in his laboratory. In another communication, dealing with the determination of traces of carbon monoxide in air, he states* that he had tried CO estimations in gas-mixtures of a dilution corresponding to that in the atmosphere of towns, by absorption with cuprous chloride or ammoniacal silver nitrate solution, and also by oxidation with permanganate and other reagents. None of these methods, however, gave satisfactory results. He was then experimenting with iodine-pentoxide at a temperature of 60° C. Some of the experiments he published a month later.† He found that CO begins to react on I_2O_5 at 30° C. The action becomes more pronounced at 40° to 45° C., and is complete at 60° to 65° C. It takes place, no matter what the dilution may be. He investigated several means of determining the amount of iodine liberated, by directly weighing it in absorption tubes filled with silver or copper filings and also by certain volumetric methods. Neither of these methods proved sufficiently accurate, and the author had resource to Müntz and Aubins' method‡, which consists in collecting the carbon dioxide formed, in KOH solution, subsequently liberating it again with sulphuric acid, and measuring over mercury. Test analyses given in detail show the accuracy of the method. Later in the same year** Gautier published a long paper on the estimation of CO in the atmosphere of Paris. In the course of it he states, that in continuation of the above-mentioned research work, he had been successful in evolving a very accurate method, capable of determining amounts of CO in the air as small as 1 in 500,000. I may here mention, that in order to achieve these results he had to use samples as large as 200 litres. This method differed from that employed by Nicloux only inasmuch as the iodine liberated from the iodine-pentoxide was caught in another tube over metallic copper, the amount being ascertained by direct weighing. The author particularly refers to the fact that the presence of certain hydrocarbons, notably ethylene and acetylene influences the accuracy of the results. He now considered 100° to 105° C. the most suitable temperature for the reaction. It is, however, noteworthy that in order to prepare the iodine pentoxide tube for weighing, he subjected it to a current of dry air at a temperature of 220° C. This is entirely in accord with the results obtained during our investigations, viz., that at temperatures up to 200° C. the iodine pentoxide, provided, of course, that it has been prepared with reasonable care, is absolutely stable.

---

[*] See this *Journal*, November, 1908, pp. 144-151.

[†] Bull. Soc. Chim. 1870, 13, p. 318.

[‡] Bull. Soc. Chim. 1889, 51, p. 163.

[**] Comp. Rend. 1898, 126, p. 746.

[††] Comp. Rend. 1898, 126, p. 793.

To those who are incredulous about the presence of CO in the atmosphere surrounding industrial centres, it may be interesting to note that Gautier found amounts of CO in the atmosphere of Paris varying between 0 and 9 parts per million. He also analysed samples of laboratory air containing considerably more.

There seems to have been a long period of inactivity after the foregoing publications, the next record of note being that of Kinnecutt and Sanford in 1900*. These authors could not obtain good results by either of the methods of Gautier and Nicloux on account of the impossibility of accurately determining the amount of iodine liberated by directly weighing it, or estimating it colorimetrically. They also stated that the gravimetric method of determining the carbon dioxide liberated during the reaction was unsatisfactory. They asserted, however, that if the iodine be caught in potassium iodide solution and subsequently titrated with thiosulphate of sodium, very close and accurate results could be obtained. Their results are in close agreement with theoretical mixtures subjected to analysis and showed that the method is capable of determining as little as 0·0025% of carbon monoxide in the air. The authors further stated that they had proved experimentally that other constituents of coal gas do not influence the accuracy of the method.

In 1903 Fillunger mentioned† that Moltersky and Nowicky employed a method similar to that just referred to, but they neglected the iodine liberated and only determined the CO_2 produced by absorption in $Ba(OH)_2$ solution and subsequent titration. The authors further asserted that hydrocarbons in the mixture had no effect upon the results. They claim an accuracy of 0·01 to 0·1% of the theoretical.

In 1905, Lévy and Pécoul‡ reverted back to Nicloux's method for the estimation of CO in confined atmospheres, but employed chloroform instead of carbon disulphide for the colorimetric estimation. In a further communication a year later,** the same authors confirmed their previous results, and asserted that small amounts of acetylene did not seriously interfere with the accuracy of the method. They found that while four parts of acetylene per thousand produce a very slight colouration, one part of CO per 10,000 produced an intense colour due to iodine liberated. They therefore came to the conclusion that in testing normal air for CO, the possible presence of acetylene may be neglected.

---

* Comp. Rend. 1898, 126, p. 871.
† Comp. Rend. 1898, 126, p. 931.
‡ Ann de l'Inst. Agron, 1881-1882.
** Comp. Rend. 1898, 126, p. 1299.

* J. Amer. Chem. Soc., 1900, 22, p. 14.
† Chem. Zeitg., 1903, p. 27.
‡ Comp. Rend., 1905, 140, p. 98.
** Comp. Rend., 1906, 142, p. 162.

In the meantime Gautier seems to have busied himself with the problem of why the volumetric estimation of CO with cuprous chloride gave such unreliable results. In a lengthy paper by himself and Clausmann,* and as a result of very carefully conducted and extensive experiments, they came to conclusions which have a very important bearing on our subject, because they not only confirm the accuracy of the I_2O_5 method, but also give specific reasons for the unreliability of gas volumetric determinations of CO in gaseous mixtures. Their conclusions may be summarised as follows:—

Firstly.—In determining CO when present in mixtures to the extent of several cubic centimetres, and after the gas in the ordinary course of analysis had been washed with KOH, bromine water, and pyrogallol, the cuprous chloride solution, even when used in considerable excess, does not absorb all the carbon monoxide.

Secondly.—CO diluted with an inert gas does not burn completely even when exploded in the presence of an excess of oxygen.

Thirdly.—The remainder of the CO escaping estimation by either of the above methods can only be determined correctly by passing it through iodine pentoxide at temperatures of not less than 70° C.

These precise statements should make it clear why the absorption method employed by some, has seemingly given negative results.

With reference to the combustion method we have never attempted the explosion with excess of oxygen, but have obtained excellent results with combustion over spongy palladium. In a synthetical gas mixture employed by us containing

1·018	vol. per cent.	CO
·867	,,	H
·767	,,	C_2H_4
4·010	,,	CO_2
1·047	,,	CH_4
19·210	,,	O
73·081	,,	N

we found after elimination of the ethylene and CO_2:—

1·02 vol. per cent. CO and 0·85 vol. per cent. H by passing the mixture through spongy palladium at a temperature of about 300° C. It will be noted from the foregoing results that the combustion is quantitative, and also that the methane present, remains unaffected.

Of the more recent publications on the estimation of carbon monoxide I can find only two worthy of notice. One by J. Tóth,† who employed the method elaborated by Kinnecutt and Sanford, for the estimation of CO in the smoke of tobacco. He filters the dried gases through cottonwool, and before passing them through the pentoxide tube contained in a glycerine bath of 60° to 70° C., he purifies them in succession with 10% caustic potash, baryta-water, sulphuric acid and quick lime.

The other publication by Morgan and McWorther* deals exhaustively with the subject. They confirm that with Kinnecutt and Sanford's method good results can be obtained. Subsequently, however, they used a modification of the method, which consists in passing the gas, after it has left the iodine pentoxide tube and been freed from all iodine fumes, through a long test tube containing barium-hydroxide solution of known strength, titrating the excess of alkali with oxalic acid, using phenolphtalein as indicator. They have proved experimentally that this method agrees well with the titration of the iodine caught in potassium iodide. They make much of the fact that lubricants used for the glass stoppers of the iodine pentoxide tube have a tendency to creep inside, thus upsetting the accuracy of the results; as furthermore, the employment of rubber or cork stoppers is, of course out of the question, they express their surprise that none of the other investigators of the subject have found it worth while to draw attention to this matter. The authors, in order to get over this difficulty, have employed and recommend the radical measure of doing away with glass stoppers altogether, by using sealed glass connections only, and there is no doubt this does away with all chances of the lubricant interfering with the reaction.

As these measures have not been strictly followed in our experiments, it might not be out of place to mention our reasons for departing from them. In the first instance, a tube with the iodine-pentoxide enclosed was not available, and we also foresaw some difficulty in removing the sublimed iodine quantitatively from the inner walls of the delivery tube. On the other hand, a spiral tube, such as Prof. Will had first of all employed for decomposing nitro-cotton,† offered very great advantages both as regards handling and working. We found that, provided the hood is clamped to the body by means of springs (see H in illustration) the joint holds absolutely tight without the use of any lubricant. The large and well ground surfaces in connection with the above arrangement being so close fitting, also effectually prevent the tendency of the oil from the heating bath to creep up. This has been proved not only by the consistency of our results and the close agreement of analysis of synthetical gas mixtures, but also by repeated "blanks," extending over periods of several hours.

---

* Comp. Rend., 1906, 142, p. 485.
† Chem. Zeitg., 1907, 9, p. 98.

* J. Amer. Chem. Soc., 1907, 29, p. 1,589.
† Z. f. ang. Chemie. 1901, p. 743.

From all the foregoing it is clear that although a great amount of research work had been carried out, there is a good deal of contradiction between the opinions of various workers, particularly with reference to the effect of certain gases, and also to the final determination of the products of the reaction. As regards the main principle of the method there is, however, only one opinion, viz , that, the reaction involved by the passing of the CO over iodine pentoxide at temperatures from 70° to 150° C., is admirably suitable for its quantitative determination, and cannot be surpassed as regards delicacy and accuracy of the results obtained. Our decision, therefore, in selecting this method for the estimation of CO in mine gases should not require further justification.

The modifications introduced and the arrangement of the apparatus finally employed by us are the outcome of numerous experiments and extensive research, covering practically all the ground traversed by previous investigators. These experiments are too lengthy and complicated to be elaborated upon in these pages. Suffice it to say that by check-analyses, blanks, employment of synthetical gas mixtures, and all other means at our disposal, we have verified the accuracy of the results obtained. As an example I will only mention that in a gas mixture containing 0·999 vol. % CO and 4·00 vol. % CO_2 and approximately known quantities of methane, ethylene, hydrogen, nitric monoxide, oxygen and nitrogen, we found 0·97 vol. % CO and 4·11 vol. % CO_2.

One other point might be specially mentioned, and that is the fact that ordinary sulphuric acid, even when as many as three absorption coils are used, does not prevent olefines passing to the iodine pentoxide tube, where, of course, they are oxidised, liberating correspondingly large amounts of iodine. We have proved this to be the case with synthetical gas mixtures containing as little as 0·251% ethylene. In the course of our investigations it has been found that, although by increasing the number of sulphuric tubes and by keeping them at a temperature of 160° to 170° C., the amounts of ethylene passing on were decreased; complete retention was only achieved by employing sulphuric acid of a strength of 99 to 99·5% H_2SO_4 and keeping the bulbs containing it at a temperature of 165° C.

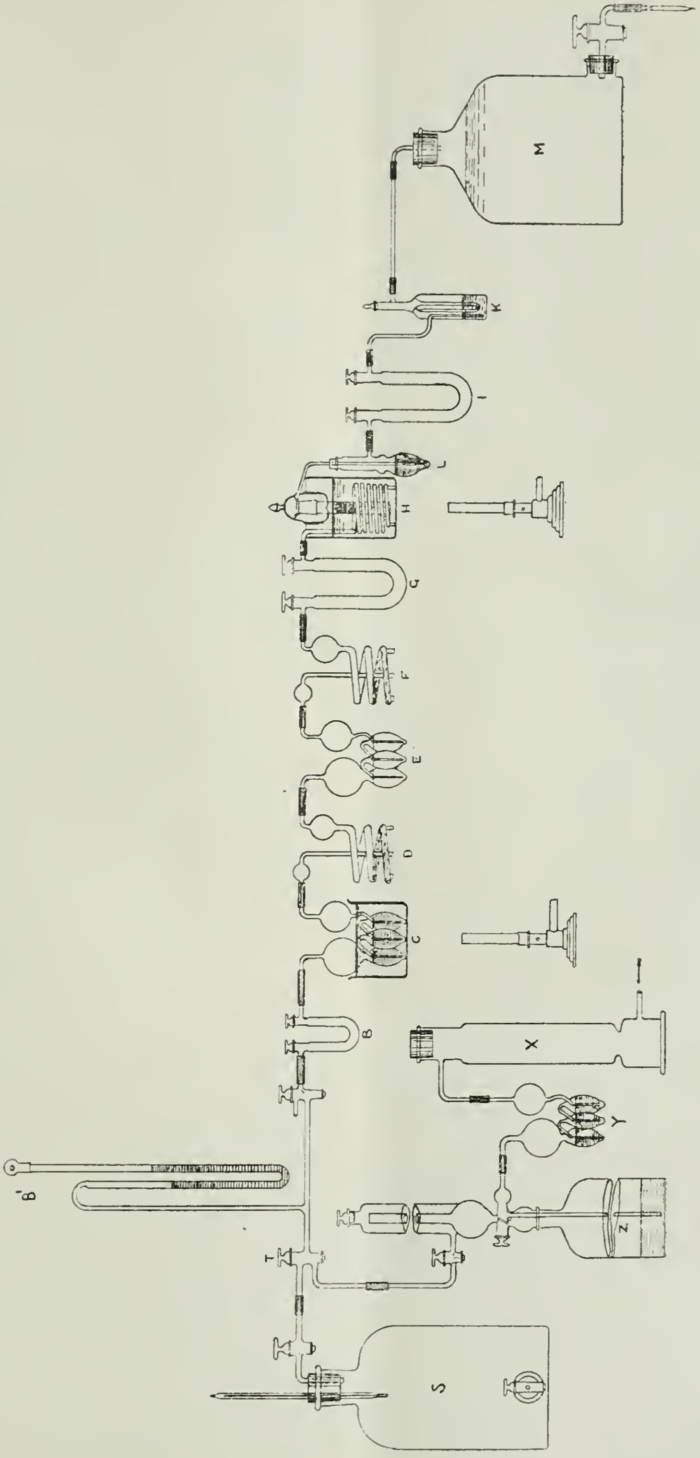
In now describing the actual apparatus, it will be observed that provision has been made to keep the pressure within it at figures very little above the normal. This state of affairs ensures against undue strain being put on the various connections and joints, and is effected by the aspirator (M), taking off part of the resistance which the gases displaced from the sample bottle have to encounter. The two-way tap (T) in front of the mercury gauge (B^1), which latter, by means of a vernier and mirror reflector, can be read to within 0·1 mill., allows of filling the apparatus with dried and purified air. The air used for this purpose has passed the chloride of calcium tower (X), the Liebig bulbs (Y) containing palladious chloride, and the washing and drying apparatus (Z), which contains caustic potash solution and soda lime. At the end of the operation, when the required amount of gas sample has been taken, the whole apparatus is again flushed out with pure air until all of the sample has been displaced.

The gases in their course to the iodine tube (H) pass first of all a U-tube (B) filled with glass wool and pumice saturated with sulphuric acid, in order to keep back organic and inorganic dust and moisture. They next pass through Liebig bulbs (C) filled with 99% sulphuric acid, kept at a temperature of 165° C, and then through the sulphuric acid spiral (D). The gases now go through the carbonic acid absorption apparatus, which consists of the caustic-potash bulb (E) the sulphuric acid spiral (F) and a U-tube (G) filled with pumice moistened with sulphuric acid. This arrangement was found necessary, as accurate results with the usual bulbs with calcium chloride tube attached, were not obtained. The iodine-pentoxide spiral tube (H) through which the gas next passes, is contained in an oil-bath at a temperature of 150° C. As will be seen the delivery tube dips directly into the absorption tube (L) containing 5 c.c. of a 15% potassium-iodide solution. The gases finally are dried in the U-tube (I) and the CO_2 formed by the reaction is caught in the small apparatus (K).

If H_2S be suspected in the sample an absorption tube containing acidulated lead-acetate solution, or a tube containing pumice saturated with copper sulphate and dried at 150° C, must be inserted in front of the first sulphuric bulb.

In cases where methane is present, we have estimated it in a porcelain tube containing copper oxide by the ordinary combustion method. This tube in such instances was inserted between the aspirator bottle and the end of the apparatus just described. With this I bring my remarks on the methods employed by us, to a conclusion.

There still remains the problem of the origin of the carbon monoxide formed. This to my mind, at the present moment can only be the subject of speculation. My reasons for saying so are, that after all we have not established very much more than the bare fact that carbon monoxide is formed in the drillhole. It is true



ARRANGEMENT OF APPARATUS FOR THE ESTIMATION OF CO AND CO_2.

that the methods of tamping and blasting generally may reasonably be excluded, not only on account of the particular care exercised during the experiments, but more so by reason of the consistent closeness of the results obtained. These would have been very much more varied if incomplete detonation or partial misfires were entirely responsible for the formation of the incomplete combustion products. This reason has tempted many to assume that the above objectionable gases originate from the explosive. Many theories based on this contention have been brought forward. It has been suggested that the nitro-cotton in the blasting gelatine does not take part in the chemical reaction involved. Others, in expecting nitric oxide as part product of an incomplete detonation, have argued that their inability to find large amounts of this gas is proof conclusive that carbon monoxide is not present either.

Counter-arguments have been advanced, that nitrogen peroxide, condensing as nitric and nitrous acid will, to a very great extent, escape being caught in the gas sample. Others again have been puzzled by the deficiency in oxygen which did not agree with theoretical expectations. A further suggestion has, I find, been made a considerable time ago by Dr. Moir*, that the cartridge paper introduced with the explosive is responsible for the formation of the products of incomplete combustion. Although we have experiments on record showing that cartridge paper, even if intimately mixed with blasting gelatine does not take part in the combustion when the mixture is exploded in a lead block, the conditions in a drillhole may be different.

Here we may have the key to the whole problem before us. The assumption that the cartridge paper takes part in the combustion would not only explain the deficiency of oxygen, the practically constant ratio of CO, and the absence of nitrogen oxide in anything like proportional quantities, but it would also furnish a reasonable explanation of the presence of hydrogen proved in the samples.

There is one more point of our subject which needs some explanation, viz., the calculation of the analytical results back to the gases formed in the drillhole. Considering the great length of my paper, I shall attempt to be as concise as possible.

I have previously mentioned that the statement of the analytical results in the above form is in my opinion the only one permissible. I have not, however, elaborated the fact that our knowledge of the chemical reaction taking place in the drillhole, on which, of course, all the calculations must be based, is by no means complete and final. The results published must, of necessity share this fate. For practical purposes they are, notwithstanding, near enough to justify their publication.

The calculations have been based on the consideration that the total volumes of CO_2 and CO found in the sample must necessarily be proportional to the total volume of gases which all the carbon in the explosive is capable of yielding. This ratio corresponds to the dilution of the products of explosion with air. After allowing for the amounts of CO and CO_2 which the air contains prior to the blast, and multiplying the analytical results by the dilution factor, the total quantities and, of course, also percentages of the explosion products are obtained.

It therefore only remains to decide on the actual amount of carbon taking part in the reaction. For this the chemical formula of the explosive involved has sometimes been taken as, for instance,

$57\,C_3H_5(O.NO_2)_3.C_{24}H_{22}(NO_2OH)_9O_{11}$, *i.e.*, that of a $7\frac{1}{2}\%$ blasting gelatine. On other occasions the cartridge paper has been stated as having taken part in the reaction, which in the case of a $7\frac{1}{2}\%$ blasting gelatine results in the formula

$$C_{210}H_{343}N_{180}O_{557}.$$

From the latter I have evolved the following reaction equation :—

$$C_{210}H_{343}N_{180}O_{557} = 180\,CO_2 + 30\,CO + 167\,H_2O + 90\,N_2 + 4\tfrac{1}{2}H_2.$$

It is interesting to compare the composition of gases so calculated with the results of an analysis of the gases produced by blasting gelatine :—

	ANALYSIS OF SAMPLE.		Calculated to drillhole gases.	Theoretical results from above equation.
	Before blast.	After blast.		
CO	·049	·55	7·22	9·85
CO_2	·356	4·64	61·78	59·11
N	79·025	75·51	28·33	29·56
O	20·570	19·12	—	—
H	—	·18	2·59	1·48

In the above calculations I have made use of figures showing the percentage composition of cartridge paper as published by Dr. Moir.*

In conclusion, I should like to take this opportunity of acknowledging and thanking Mr. Greig for the able work and assistance which he has rendered during the whole course of these investigations.

They are not yet completed, but the results obtained so far have, I trust, been of some assistance in establishing the fact that CO in considerable quantities is present in the drillhole gases of blasting gelatine.

---

* See this Society's Proceedings, vol. iv., p. 46.

* See this *Journal*, vol. vi , April, 1906, p 307.

During the course of the reading of the paper the author demonstrated by means of the apparatus (see illustration) the method of analysis adopted.

The President: I am sure we are greatly indebted to the author for his most interesting paper, and I beg to move a very hearty vote of thanks to him for the trouble he has taken in compiling it, and for the demonstration which he has given us.

Dr. J. Moir (*Vice-President*): It is with great interest that I second this vote of thanks to Dr. Weiskopf. He has done a great service in collating all the literature in connection with this delicate method of analysis, and in applying all the improvements to his own apparatus. I, for one, had no idea that the I_2O_5 method had been known so long. By the way, he omitted to mention the paper of Thorpe, the English Government Analyst,* which came out whilst I was engaged with the Miners' Phthisis Commission analyses, and enabled me to do a few accurate determinations in mine air before the close of that work in 1903. I soon found out that lubricants must be avoided; and I have used a U tube with side tubes, first filled with I_2O_5, and then drawn out and sealed off and finally purified by drawing dry air through it at 190° for several hours.

As for olefines, I freely admit that the evidence for them in mine air is very weak; however, I tried the experiment of aspirating a large volume through bromine water, and on treating with alkali got distinct signs of an ethereal di-bromide, together with a solid bromo-compound apparently connected with the hydro-carbons developed from the burnt guttapercha of the fuse.

As for the origin of the CO found after blasting, I am very pleased to find the author accepting my views of long ago,† that it is due to the wrappers of the cartridges. It is gratifying to be believed at last, even when one is voicing opinions not to be found in the last generation's text-book! I was not believed in 1903—I suppose I was then a dangerous scientific firebrand, "running amok" amongst rule-of-thumb pseudoscience, and the curious preconceived ideas which characterise the "practical man."

Mr. W. Waters (*Associate*): I should like to ask Dr. Weiskopf just one question on a point which he seems to have omitted to refer to. He suggests the possibility of the combustion of the cartridge wrapper being responsible for the constant ratio of carbon monoxide. This can easily be ascertained, as holes can be charged with explosives with the cartridge wrappers taken off—just with the naked blasting gelatine. I should be glad to learn whether any experiments have been made with gelatine with the cartridge wrappers stripped off.

Dr. Weiskopf (*Member*): I do not at present propose to go fully into the questions which have been put, but will deal shortly with some of the points raised. With regard to Mr. Waters' suggestion to sample the gases obtained by exploding blasting gelatine without the cartridge paper in the borehole, this is one of the experiments I had in mind, when saying that our investigations are not yet completed. We all realise the great importance of the subject under discussion. The miner is greatly interested in it from his own point of view, but we, as manufacturers, also have to consider the stigma which possibly may remain with the explosive. As Mr. Cullen intimated at a previous meeting, everything possible will be done to improve the explosive in this respect, should it be proved that the explosive itself is to blame.

As regards Dr. Moir's query *re* olefines, we have found that these are kept back completely by sulphuric acid of $99\frac{1}{2}\%$ H_2SO_4, provided the bulbs containing it are kept at 160° C. The other point, namely, how the amount of gas sample taken is ascertained, will be found fully described in the appendix to my paper.

Briefly, during the analysis a weighed amount of mercury is gradually introduced into the sample bottle. At the end of the estimation the pressure inside the bottle is so adjusted that the remainder of the gas sample is again under the original pressure and temperature conditions, previously determined. Thus the number of cubic centimetres of mercury introduced corresponds to the number of cubic centimetres of gas displaced from the sample bottle.

The President: I ask your indulgence to allow me to leave the meeting, and I will ask Mr. McArthur Johnston (Vice-President) to take the chair. The reason is that I have been invited, as President of your Society, to represent you at the dinner of the Mine Managers' Association.

The President then retired, and Mr. A. McA. Johnston took the chair, and called upon Prof. Wilkinson to read the following paper:—

THE DISTRIBUTION OF THE GOLD PRODUCED ON THE RAND.

By T. Kirke Rose, D.Sc., A.R.S.M., M.I.M.M., F.C.S. (Hon. Member).

By common consent, statistics are admitted to be both misleading and dull, and the following account of the first rough distribution of the gold produced in South Africa cannot be expected to escape the common lot of such papers. Consequently I beg the Society to remember that the paper was written at the invitation of the Council, following a suggestion made by Mr. T. L. Carter. In drawing up this account I have used official publications only and have everywhere reached the limits of information almost at the beginning of the enquiry. My own private quarrel with statistics is, not that they are misleading and dull, but that they always stop short before they have told me what I want to know; so I am continually driven to making guesses.

---

* *Journal Chem. Soc.*, London, 1903.

† See this Society's Proceedings, vol. iv., p. 46, and this *Journal*, vol. vi., April, 1906, p. 307.

The official publications which have been consulted are the Annual Statements of Trade of the United Kingdom, the Statistical Abstracts of the United Kingdom, and of the British Colonies, Possessions, etc., and the Annual Reports of the English, French, and United States Mints.

I have considered only the period of five years, 1903—1907,* because during the war the production of gold in South Africa was comparatively unimportant, and the years 1899–1902 were abnormal in that and other respects. In a limited sense every year is abnormal. The conditions affecting the distribution of gold change continually, and it is useless from a study of statistics alone to hazard any prediction as to the future.

At the beginning of the study of what becomes of the Rand gold, it must be premised that no distinction can be drawn between the gold from the Rand, and that from other districts. The gold from the Rand goes down to the coast and is at once, statistically speaking, mingled inextricably with that from the rest of the Transvaal and from Rhodesia. The combined stream flows into London, and there joins other streams, and its identity is lost again. Little rills flow out hither and thither and are caught up in other currents, so that by imperceptible degrees the whole world is involved. In this paper the assumption has been made throughout that wherever two streams unite and become indistinguishable from each other, they are at once completely mixed, so that any sample, however small, contains the same proportion of each constituent as the whole mass.

When the Rand gold leaves South Africa, a slight difficulty is at once encountered, for no definite statement is available as to its destination. The declared value of the production of gold in the Transvaal and Rhodesia was, for example, £26,615,025 in 1906. The value of the exports of gold from British South Africa is given as £26,516,475 for 1906 in the Statistical Abstract. In the "Annual Statement of Trade of the United Kingdom" the value of the gold bullion imported into England from South Africa is not given, but only the gross weight in ounces, *e.g.*, 6,986,832 oz in 1906. The result is that the imports into the United Kingdom cannot be identified with the exports from or the production of British South Africa. These three figures are, however, in sufficient agreement (taking the mean fineness of South African bullion imported into England as 875 in gold) to justify the assumption that practically all the South African gold is exported to London. Apart from the figures, this is, of course, perfectly well known.

On arrival in London the South African gold is mixed with bullion coming from other parts of the world and can no longer be distinguished from it. This matters the less for the reason that the South African bullion forms the bulk of the total imports of gold bullion, as is shown in the following table, which is based on a table in the "Annual Statement of Trade for 1907":—

IMPORTS OF GOLD INTO THE UNITED KINGDOM.

Country whence exported.		Year 1905. oz. troy (gross)	Year 1906. oz. troy (gross).	Year 1907. oz. troy (gross).
British South Africa ...		5,833,741	6,986,832	8,012.437
British India		616,320	571,033	547,452
Australasia...		972,342	729,237	803,961
Rest of World		664,828	805,703	1,334,608
Totals	Quantities	8,087,231	9,092,805	10,698.458
	Value £	30,114,369	33,865,361	39,956.304
Fineness of bullion ...		876·6	876·8	879·2

In these years South African gold is seen to form 72%, 71%, and 75% respectively of the total weight, and judging from unofficial returns, the values were in nearly the same proportions. The bullion on arrival in London is in part refined there and in part re-exported in an unrefined state. Beginning with 1908, the Customs Establishment will publish statistics distinguishing between unrefined and refined bullion imported and exported, but no figures are available as yet. My own opinion is that some 15% of the imported bullion was re-exported in the period 1903–1907, without being refined in London. Both the amount and the percentage of such bullion appear to be increasing, but there is little to go upon in forming these opinions. The refined bullion is partly retained in the country and partly exported.

---

* Difficulties with this period were encountered owing to the fact that the latest United States Mint Report issued is for the year ending June 30, 1906, and the latest French Mint Report is for the year ending Dec. 31, 1906. Some of the figures given in these reports extend only to the end of 1905. The Statistical Abstracts come down to the end of 1906 only.

Comparing the total imports and exports of bullion, we get :—

	1903–7.	1907.
Imports ...	£151,973,235	£39,956,304
Exports ...	£98,059,568	£26,589,442
Excess imports ...	£53,913,667	£13,366,862

Here the imported bullion is divided clearly into two parts, one part required in the United Kingdom and retained there, the other part not required and re-exported without delay. These will now be considered separately.

Bullion Retained in the United Kingdom.—Dealing first with the bullion which is retained in the United Kingdom, it is at once evident that most of it is converted into coin, thus :—

	1903–7.	1907.
Coinage from ingots only at the Royal Mint, London ...	£48,944,370	£16,098,894

Some of the ingots received for coinage, however, are derived from the melting of gold coin. No exact record is kept of the value of these, but, in my opinion, it may be put at the following amounts :—

	1903–7.	1907.
Ingots resulting from melting gold coin ...	£1,392,000	£4,500

The coinage from refined ingots at the Royal Mint has accordingly been approximately as follows :—

1903–7.	1907.
£47,552,400	£16,094,400

The rest of the bullion retained in England but not used for coinage, which amounted to about £6,361,300 for the five years 1903–7, was added to the stock of bullion already in existence, or was used in the arts, for the manufacture of jewellery, for gold plating, for the manufacture of gold leaf, in dentistry, in photography, etc.

The stock of bullion in the United Kingdom, apart from the floating stocks held by refiners and bullion dealers, is in the vaults of the Bank of England. No figures are available as to the amount so held, the Bank making known only its stock of bullion and coin together. The stock of bullion is, of course, a fluctuating amount, but any gradual permanent change is doubtless in the direction of an increase of the stock. The stock of gold bullion and coin held by the Bank on Dec. 31, 1902, was £27,840,385, that on Jan. 2, 1907, was £29,270,470, and that on Jan. 1, 1908, was £31,602,515. For purposes of comparison, it may be remarked that the mean stock in the first week of January of the ten years 1881-90 was £20,550,000. It is likely enough that a part of the increase shown here was in the form of ingots, but it would be pure speculation to hazard a guess as to the exact amount. During the year 1907, it is quite possible that the stock of bullion was reduced.

Leaving all this on one side therefore, we have the sum of £6,361,300 in ingots, which was all or for the most part used in the industrial arts in the United Kingdom during the years 1903-7. No estimates of the consumption of gold are published officially in England, though the matter will doubtless be dealt with in the forthcoming Census of Production. The total consumption for the three years 1903-5, however, has been estimated by the Director of the United States Mint* at £7,706,000, excluding old material (remelted jewellery, etc.), but including gold coin. This seems to me, judging from those returns of the manufacture of jewellery to which I have had access, to be a reasonable figure, but the annual consumption in 1906 and 1907 was certainly higher than in 1903-5. The amount of English gold coin melted annually for use in the industrial arts is estimated at £500,000 at least,† its only known use being for the manufacture of wedding rings. This would leave an annual amount of about £2,000,000 to be derived from gold ingots or £10,000,000 in five years. There is here a discrepancy of nearly £4,000,000, which is most readily accounted for by assuming that the amount of British gold coin melted down for use in the arts was underestimated in 1903, and that it really amounts to £1,500,000 per annum. The disappearance of immense quantities of British gold coin has certainly never been fully and satisfactorily explained. The discrepancy would be equally well accounted for by the assumption that £1,000,000 in gold coin is melted down annually in the United Kingdom and subsequently exported.

Bullion Re-exported from the United Kingdom.—The destination of the exports is shown in the following table, which is abstracted from the Annual Statements of Trade for 1903—1907 :—

Countries to which exported.		Period 1903–7. oz. troy (gross).	1907. oz. troy (gross).
Continent of Europe		13,005,963	2,021,561
United States ...		6,063,671	3,143,700
British India ...		4,630,381	1,374,879
Rest of World ...		227,822	1,272
Totals	Quantities	23,927,837	6,541,402
	Values ...	£98,059,568	£26,589,442
Fineness of bullion		964·8	956·9

---

* The figures are taken from the Annual Reports of the Director of the French Mint. They are stated by him to be quoted from statistics published in Washington on the authority of the Director of the United States Mint, but they do not appear in the Annual Report of the United States Mint.

† Annual Report of the Deputy Master of the Royal Mint, London, 1903, p. 22.

When expressed in percentages the above table becomes :—

EXPORTS OF GOLD BULLION.

Countries to which exported.	1903-7. %	1907. %
Continent of Europe ...	54·4	30·9
United States	25·3	48·1
British India	19·3	21·0
Rest of World ...	1·0	—
	100·0	100·0

Concerning these tables it may be observed that "Europe" stands chiefly for France and Germany, which together take almost the whole of the exports in certain years. There are exceptional years, however, as for example the year 1906, when 402,559 oz. went to Russia, as against 1,140,537 oz. to France, and only 77,068 oz. to Germany. France took 7,674,007 oz. in the year 1903—7, and Germany 3,981,192 oz. in the same period.

The gold bullion imported into France for the most part stays there, the exports of ingots being inconsiderable. The bars from England form over 30% of the total imports of gold and over 40% of the imported ingots. About one-half of the imported bar gold is coined, chiefly into 20-franc pieces; about a quarter of it is employed in the industries; some 5% of it is re-exported, and the remainder presumably goes to swell the stock of bullion in the Banque de France, which increased by about £16,000,000 in the four years 1903—6.

Of the gold bullion imported into Germany, something like 20% is re-exported, another 20% is used in the industries, and the remainder is coined.

The gold ingots sent to America have varied in amount more than those exported elsewhere. From 1903 to 1907 (the year of the panic) they increased steadily from 158,945 oz. to 3,143,700 oz. There was, of course, a large reduction in 1908. The mean for the five years 1903—7 amounted to over 10% of the gold deposited at the United States Mints. Of this total quantity, in the years 1903—6, about 55% was coined, about one-seventh was used in the industrial arts, and the remainder was added to stock or exported. According to the United States Mint Reports, however, the ingots exported are almost invariably United States fine bars, practically never foreign ingots. Also rather more than 40% of the imported gold bars were issued for use in the industrial arts in the years 1903—5. This would leave nearly 60% of the imported bars to be divided between coinage and additions to the stock of bullion in the United States Mints and Assay Offices. This stock is subject to great fluctuations, *e.g.*, it was $124,083,823 on June 30, 1902, $51,910,029 on June 30, 1905, and $104,383,158 on June 30, 1906. No exact information is available as to the stocks on January 1, 1903, and December 31, 1907, and it is better to leave this on one side, as in the case of the stock of bullion in the Bank of England, and to assume that 60% of the imported bars were used for coinage. The difference between the percentage distribution of the total deposits in the United States Mints and the distribution of the imported gold bars (of which those from England form only a part) illustrates the difficulties of this part of the enquiry, and shows how great may be the errors in the statements as to the disposal of the gold exported to France and Germany.

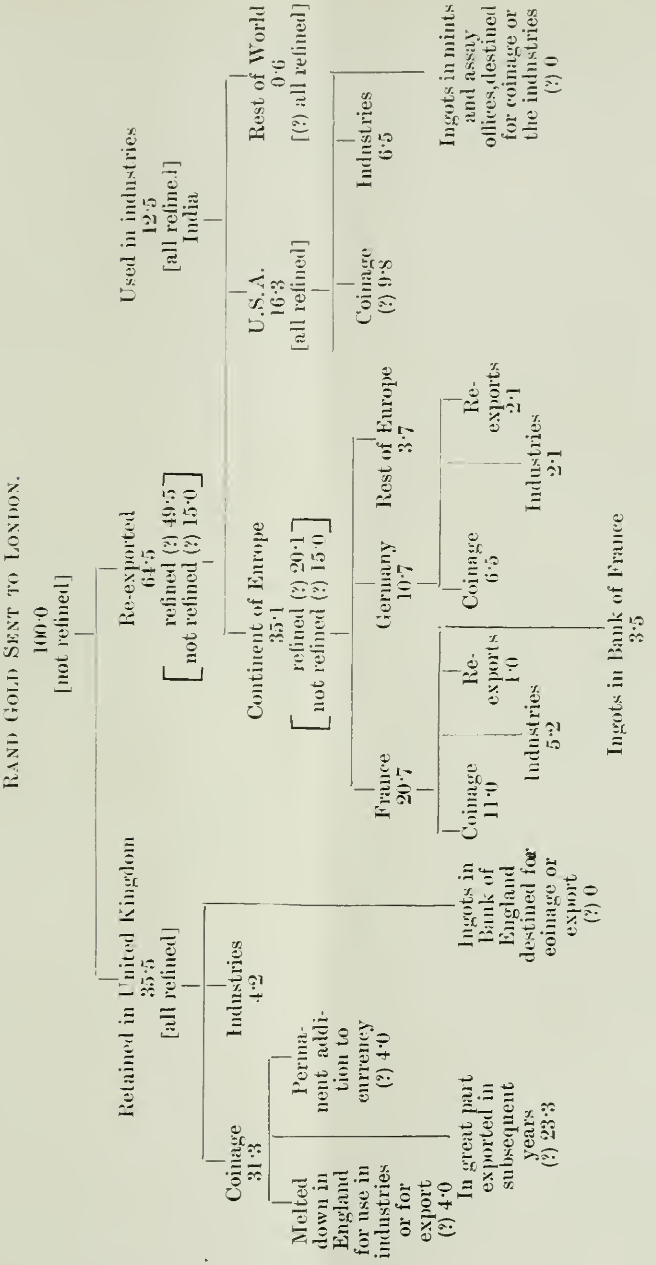
The gold bullion (chiefly or entirely in the form of 10 oz. fine bars) which is sent from England to India, disappears on arrival in that country and is probably in great part converted into jewellery. As the Director of the United States Mint remarks, "The tide of gold and silver has been flowing into India for centuries." Probably some large hoards exist, but, no doubt, there are immense numbers of small hoards. From 1835, when the official records begin, the net imports to the end of 1906 amounted to £193,880,000, in addition to the production in India, which was £23,400,000. The world's production for the same period is estimated at £1,920,000,000.

The wanderings of the gold bullion from the Rand have now been traced as far as is practicable. I have drawn up the results in the form of a percentage chart which may be of interest, based as it is on the facts and considerations set forth in this paper, although it is not without its fanciful side.

Summarising the chart, one gets for the disposal of the Rand gold in the period 1903-1907 :—

Used in coinage - -	58·6 per cent.
Used in industrial arts -	30·5 ,, ,,
Deposited as ingots in the Bank of France -	3·5 ,, ,,
Undetermined - -	7·4 ,, ,,
	100·0 per cent.

The result may also be put in the form of a bet, such as that of any particular ounce of gold produced on the Rand, it is 7 to 1 against its disappearing into India as refined 10 oz. bars, 99 to 1 against its being exported from France to an unknown destination, and 69 to 31 against its being coined at the Royal Mint. In the latter event its history could be followed further, though with increasing difficulty, but the adventures of a British sovereign are too varied to be described at the end of a paper which is already far too long.



The Chairman: We have had a very interesting and instructive paper. It goes further into the subject than we here have been in the habit of doing, and lets us know where our gold disappears to. Unfortunately it does not seem to come back here fast enough. I propose a hearty vote of thanks to Dr. Kirke Rose.

Mr. H. A. White (*Member of Council*): I should like to second the vote of thanks to Dr. Kirke Rose for his able and interesting paper, and to call attention to the steady increase in the fineness of the gold bullion exported from this country. Part of this increase is due to efforts of members of this Society in that direction, no doubt, but I am not certain that part is not also due to improvement in the method of settling assays, whereby a nearer approximation is got to our own local results. In connection with that last bet referred to by the author, I would like to see the odds against the mint increased by the establishment of a similar institution for South Africa, perhaps that may yet be counted amongst the benefits of closer union in the days to come.

The vote was agreed to.

The Secretary read the following reply :—

WESTRALIAN WET-CRUSHING PLANTS, WITH SOME NOTES ON LABOUR EFFICIENCY.

(*Read at February Meeting, 1908.*)

By GERARD W. WILLIAMS (Corresponding Member of Council).

REPLY TO DISCUSSION.

I must apologise for the delay which has occurred in regard to this reply. The first numbers of the *Journal* containing reports of the discussions went astray, and when duplicates were sent out to me I was somewhere "off the map," in Northern Queensland.

I am afraid that I must plead that I have been "misunderstood" by some of my critics. I had no intention whatever of advocating white labour in place of Kafirs for doing legitimate unskilled work, but I submit that none of my critics have said a word in contradiction of my statement, "the skilled worker on these fields does his work on his own, whereas on the Rand he almost invariably has the assistance of one or more natives." Far be it from me to suggest that a "white labour" policy would be practicable on the Rand, even a casual acquaintance with Westralian or Victorian mining entirely disabuses one of that idea, but I still maintain that the skilled white worker should do as much work in Africa as he would do on other fields, and I still maintain that no small number of boys are employed on the mines whose work is wasted in that they simply serve to relieve the skilled worker from legitimate exertion.

To turn to individual criticism : Mr. H. A. White's remarks have been practically answered by Mr. Brett, whose acquaintance with Kalgoorlie is infinitely greater than mine, and who is particularly well qualified to speak on behalf of the field with which he was formerly associated.

Mr. White's statement that the Horseshoe costs lend small support to "the idea that all-white labour could be used successfully on the Rand" was the last remark that I expected from him. Surely, from the itemised costs sheet which I published, it is obvious that there can be no direct contrast between the involved processes of the Horseshoe and the comparatively simple operations on the large plants of the Rand. Incidentally, the Horseshoe's new mill is entirely all-sliming. Neither do I advocate the "continuous process" for the Rand. Far from it. But I do consider that having in view the nature of the Horseshoe ore that the results (a total recovery of 88·4%) are good, and especially so when one remembers that a few years ago the Kalgoorlie residues averaged 3 to 4 dwt.

The highest recoveries are obtained on the all roasting plants. A table showing the work done, recoveries and costs on the leading dry-crushing plants, is appended.

My remarks, "applied with equal success under alien conditions," referred to the attempt made to run certain Rand plants on the Westralian "continuous process" system. The attempt, disguised under the name of the "New Metallurgy," has been much discussed by the Society, and needs no further mention. Of course, "equal success" is a relative term, but it is impossible to deny to the Westralian metallurgists the credit of having brought the treatment of complex sulpho-telluride ores by wet-crushing to a successful issue.

I entirely concur with Mr. White's remarks *re* the general inadequacy of the tube mills of the West. They are far too small, and it is certain that it would pay to replace them with larger machines. Touching the question of the duty in the Sons of Gwalia mill, I can only say that the figures quoted are official, and that I have every reason to believe that they are accurate.

All ore delivered to the mill passes over a weighing belt. I have not inspected the plant personally, but I am certain that the data are correct. The ore is certainly not so "tough" as the Kalgoorlie ore. I admit freely that inside amalgamation belongs to the past, but in mines where much of the ore occurs in an extremely coarse form, or where, rather, there is always the possibility of "specimen stone" passing into the battery, there is certainly something to be said for the idea. Personally, I dislike it and never use it, as I think that the loss of floured quicksilver is often considerable. Where concentrating tables follow the mill this loss is reduced to a minimum.

To Mr. F. F. Alexander I would remark that I did not "recommend for his notice" such "time honoured abuses as inside amalgamation, grinding pans and 3 ft. 3 in. tube mills." I wrote a description of Westralian metallurgy and these practices obtain there. I assure Mr. Alexander that I am in no wise responsible for them, nor do I desire other fields to copy them. The Kalgoorlie field has evolved itself, and all existing plants represent the more or less complete result of such evolution. In few cases have plants been rebuilt, they were remodelled and older apparatus was utilised. And whatever the grinding pan may do in other parts of the world, it is certainly a success in Kalgoorlie where the tough ore is more amenable to grinding by attrition than by impact.

"A numerical increase of the total employees on Westralian plants would tend both to reduce working costs and improve extractions." This may be interesting as an expression of opinion on the part of Mr. Alexander, but as it is entirely unsupported by fact it is of little value. Personally, I have not found the plants undermanned.

Mr. Alexander must remember that they are manned by whites who are all, more or less, skilled, and whose efficiency is extremely high. In the next sentence my critic goes on to state that the irresponsible Kafir is actually a handicap to the skilled worker. Doubtless, and if so, why employ him except for absolutely unskilled work? Surely, this is the strongest possible argument in favour of my contention, that the skilled white should do his own work.

I thank Mr. Brett for taking up the cudgels on behalf of the Westralian metallurgists; the small tube mill is certainly less efficient than the large ones used on the Rand, but it is far from being the toy that Messrs. White and Alexander imagine.

I admit freely that Mr. Weston is correct in calling me to order for not having stated that wood fuel was exclusively used in the West, and that its use renders stoking much easier work. But the fuel used does not affect the engine driver or diminish his task of keeping the engine room going without assistance. Again I would ask, how many boys would be allotted to that driver if he were on the Rand? Certainly two per shift.

In conclusion, I wish to disown any attempt or desire to deprecate Rand metallurgy. I assure members that I have ever battled hard for the reputation of my "mother camp;" indeed I may have made myself not a little disliked by my persistent advocacy of certain Rand methods. But on the one point upon which I ventured to criticise Rand practice, the labour question, none of my critics have brought forward any serious rebuttal of my statements. I am somewhat disappointed in this. I half hoped that the criticism would be a little severe, and I hoped that some definite refutation of the charge of employing native labour simply to save a white skilled worker from legitimate exertion, would be forthcoming. The employment of native labour in South Africa is an economic necessity both from a mining and a national point of view. But such natives should only be employed on unskilled work. The employment of natives for the purpose of relieving a skilled white worker from the manual work properly associated with his craft is not only uneconomical; it is a practice fraught with real danger both to the individual and to the community.

I am writing this towards the end of the year. I beg to send my former colleagues and members the best of good wishes for the New Year. "If the wanderer in the ends of all the earth" may seem to some a little critical in regard to his old camp, he would assure them that he is not forgetful of what he owes to the field at large, and the Society in particular.

EQUIPMENT OF ALL-ROASTING PLANTS, KALGOORLIE, W.A.

Mine.	Monthly Tonnage.	Value per ton (2,000 lb.)	Primary Crushing Plant.	Roasters.	Secondary Grinding Pans. Number.	Secondary Grinding Pans. Diameter	Recovery per cent.	Costs Treatment. s.	Costs Treatment. d.
		dwts.				ft.		s.	d.
Great Boulder ...	14,300	17	3, No. 8 Krupp Ball mills	13, Edwards	12	5	91·0	11	6
Perseverance ...	17,500	7·5	12, 30 in. Griffins 16, No. 11 Griffin	8, Merton 6, Edwards (Perseverance type)	14	8	91·5	11	3
Kalgurlie ...	10,000	15·0	9, No. 5 Krupp	15, Edwards	15	5	94·5	11	10
South Kalgurlie	9,000	7·0	2, No. 8 Krupp 1, No. 5 Krupp	8, Merton	5	8	94·2	12	9
Associated ...	9,900	10·0	10, No. 5 Krupp	18, Merton	20	5	90·0	11	1
Associated North	3,500	19·0	3, No. 5 Krupp	5, Merton	8	5	92·5	12	1

REMINISCENCES OF THE EARLY RAND.

(*Read at August Meeting, 1908*).

By M. H. Coombe (Member).

DISCUSSION.

Mr. S. H. Pearce (*Past-President*): It is not an easy task to carry one's mind back some 20 years to obtain anything interesting with regard to one's early experiences on the Rand, and I am afraid the general impression I have of my earliest days here consists chiefly of homesickness, accentuated by the usual discomfort of living inseparable from a young mining camp, and which told severely upon my far from robust constitution at the time.

One of the first things that I remember about this time was sharing a room in the then best hotel which, by the way, consisted of a long iron building, partitioned into rooms with calico and paper, and being considered fortunate at not having to sleep on, (or under—if it rained), the billiard table. My first room mate was a very friendly and talkative young man, who cheered me up by telling me that the Newbery Vautin process, which was responsible for my coming here, was foredoomed to failure, as he was out in this country for the purpose of ascertaining the suitability of the "banket" to a new process he was connected with, and that he was satisfied that it would supersede all others in dealing with battery tailings. His statement was prophetic, as the young man turned out to be Mr. Alfred James, now President of the Institution of Mining and Metallurgy, who took a prominent part in introducing the cyanide process. A few days later I obtained permission to start an assay office in order to occupy my time until the projected works were built, and so my principals acquired the premises, stock, and goodwill of an established business, as a going concern. I never knew my predecessor, and took the goodwill for granted. The premises consisted of a small room in the backyard of a now flourishing bank, and the stock I can almost itemise from memory, as it comprised very few things. The principal items were, outside:–a pestle and mortar, prospector's pan and a half tub of water; inside:–a Morgan "A" muffle furnace, a few tiny crucibles, a cupel mould, and tins of bone ash, soda carbonate, borax, litharge, etc., and two pairs of tongs. A pair of apothecary's scales appeared to have done duty for both weighing charges and results, and I have often wondered if I should do him an injustice by supposing that the bulk of his work was done in the outside premises. Fortunately however, a complete outfit had already been ordered, and I was able shortly afterwards to establish myself in more commodious premises, on the site where the extension to the Rand Club now stands.

It was about this time that the "blue bar" scare took such a hold on the capitalists of the time, which came about in this way. The principal technical mining experts of the day had pronounced the reef to be alluvial—probably I suppose on account of the pebbles which gave it something of the appearance of gravel. When the shafts reached the pyritic zone, the change in appearance, together with the fact that the reef was generally poor at that point, led them to the belief that "bottom" was reached, and they declared it was the so-called "blue bar" or granitic floor of the alluvial they were accustomed to. One of the few points at which this level was reached was on the Meyer and Charlton mine and a sample was given me, assaying only 6 dwts., which compared very unfavourably with the *ounces* they recovered from the surface ore. Strangely enough, this sample showed practically no gold in the pan, and one of my tasks was to demonstrate its existence, as much scepticism was evinced at my assay certificate. In fact, I had eventually to allow the doubter to carry out the operations himself, of panning to clean concentrates, roasting and regrinding in an agate mortar to free the gold, before he was convinced that I was not guessing.

I cannot recall anything of particular interest in regard to the introduction of the cyanide process; being then interested in another form of recovery, I do not suppose that I was over anxious to believe that the results claimed were being actually obtained. My first experience of its possibilities was, however, very striking. I was asked to prepare a trial lot of clean concentrates for the Gold Recovery Syndicate with the idea of ascertaining if the process was capable of treating the pyritic ore. To this end I negotiated the purchase of 1¼ tons of vanner concentrates from the Percy Gold Mine (now part of the Treasury G. M. Co.), assaying about 24 oz. per ton, which were duly delivered. The importance of the test was considerable, and an idea of the scepticism regarding the process can be formed from the fact that Mr. (now Sir) A. Woolls-Sampson and myself accompanied the parcel and watched every detail of the trial until the residues were ready to sample; even going as far as to roll ourselves in our blankets and sleep alongside the vats. Speaking entirely from memory, I believe the trial occupied about 36 hours. The dissolving of the gold occupied about 24 hours, and this operation was carried out in a paddle agitator tank with a solution of about 5% KCy. Samples were taken at intervals to ascertain the progress of solution, and the charge was then run into a leaching vat for washing; this operation was assisted by a steam vacuum and lasted about 12 hours. There was a good deal of discrepancy in the assay result of these residues, as the pyrites seemed to trouble most of the local assayers, who produced low results. They were finally agreed to contain 9 dwts., and thus an extraction of over 98% was certainly obtained, and disposed of any doubt I previously had in regard to the capabilities of the process. There is another point I might mention, to show that distrust did not exist entirely upon one side, viz., that I was thanked by Mr. James for supplying clean concentrates, and for not introducing material which would detrimentally affect the trial.

Mr. Darling has given you a very valuable and detailed account of the results obtained from the Robinson tailings plant, which may be considered as typical of the time, as regards the value of material treated. In 1891 I went to the Nigel mine, where the Pearce-Bowness Syndicate was treating its old tailings, which had a value of over 4 oz. per ton. Residues of 10 and 12 dwt. were quite usual and accepted quite cheerfully, for the profits were very big, although the costs often ran up to £1 per ton treated. It is, however, interesting to note that at one time there was some serious consideration of turning this plant into a dry crushing, roasting and chlorination concern, which, under the existing circumstances, appeared to be quite justified for there were some 20% of the tonnage crushed discarded as slimes, in addition to these residues. One sometimes wonders among the many "ifs" that cross one's mind, what would have been the outcome of this being actually done and the system extended? That is, to what pitch roasting and chlorinating could have been brought, if all our energy and research had been expended upon this, instead of upon the cyanide process. Of course, these are idle thoughts, but when one thinks of the ignorant groping work we all did in those first days, and the plant and material we had to work with, there is room for much reflection.

It is at such times that questions will arise as to the reasons for the indifferent work then done. We are not all of us in a position to show such excellent results as those of Mr. Darling; and most of us will have to confess that deficiencies in actual recovery were only counterbalanced by getting as big a ton as we could. As a matter of fact, in view of our present knowledge, it seems incredible that we should have obtained the results shown on paper in those days. Let us recall a few of the conditions which then existed. Firstly, the capacity of the plant was reckoned upon a three days' treatment as a rule; during which time a preliminary wash, strong solution, and weak washes, followed by a water wash, were all supposed to be given to each charge. We can imagine what the values of last drainings were; the determination of which was a refinement left for much later investigation. Secondly, we had that "special" cyanide carrying 65% KCy, the balance being potassium carbonate, caustic potash, and a large quantity of that carbide, which was such an excellent precipitant, and responsible for the ounces that were cleaned up frequently from sump sediment. Thirdly, we had precipitation so haphazard in character, that cases have been known where solutions containing at least half an ounce were used to wash charges containing a far lesser value. It was a case of, "there were the boxes full of zinc: If the gold would not come down—well, it would not"—that was all. Fourthly, there was the sand itself—to be taken from the storage dams. Circumstances were kind to some, who had large quantities of clean oxidised sand to draw from. Others were not so favoured, and had to take it as it came, whether it was acid or not, with or without varying quantities of slime, which often made nice sloppy tanks to discharge. Lastly, we have to give the millmen away, and record that amalgam was a frequent constituent. Many were the high residues from this cause that were reckoned as salted, and never recorded in our average. I have not included the fineness of crushing in this list. Every millman had his own fancy in that respect, and grading analyses were not dreamt of. It is quite good to remember these things, and the general reticence of cyanide managers as to their methods. One could go on almost indefinitely with such rambling reminiscences, but I think it would be appropriate to stop here, and reflect that this Society can congratulate itself in the part it has taken in overcoming this reticence, and in developing the cyanide process to its present stage, and in so doing, be proud that it has contributed so much to the making of the world's greatest goldfield.

REPLY TO DISCUSSION.

Mr. M. H. Coombe (*Member*): I wish to express my thanks for the interest displayed in my paper, and my reply will not be difficult to make. Its main object will be to thank those members who gave such interesting elaboration of the subject. The President for his photo of the old Wemmer and old Johannesburg; Mr. Curtis for his reminiscences of the Market Square, etc.; Mr. Caldecott for his extremely interesting contribution; Mr. MacArthur and Mr. Darling for the account of their early metallurgical struggles; and Mr. Cotter for the humorous recital of his experiences in Barberton and on the Rand.

Mr. Curtis challenges my statement *re* wages paid to whites and natives. In Sept., 1887, I was engaged to open up the Aurora G. M. Co. claims, not at 50s. per week, as I am made to say in my paper, as printed in the August number of the *Journal*, but at 65s., which was then 5s. more per week than the highest wage. In the previous April or May of that year I was offered work on the property then known as "Whitehead's," now the Crown Reef; at 50s. a week, and this for a specially skilled job. They had sunk a small shaft in the vlei and were driving out under to tap water, I think in preparation for a mill. The ground was clay, very wet, and had to be close-timbered and "spiled." I went down the ladder-way with the foreman, had a look at what required to be done, asked £3 a week for the job, was indignantly refused, and left.

Of course, as work developed, the demand for men greatly increased, and wages rapidly became higher, but I am sure that the prevailing rates in 1887 did not run higher than £3 per week, and the average was probably about 50s.

Natives at that time were very scarce. In June of that year I secured a contract from the manager of the Branston G. M. Co., and had to pay 20s. a week for Kafir labour.

I cannot pass by Mr. MacArthur's contemptuous references to the "Practical Man" without a protest. In the days of which he speaks, and of which I wrote, the clerk, storeman, sugar planter, stock broker, mine manager: whom force of circumstances had placed in charge of a mine, in most instances, unfortunately for the mine, had to do exactly what the "well trained scientific young fellows" have to do to-day, that is, lean on and off load their responsibilities on to, the shoulders of one or more of these "Practical Men." I am sorry Mr. MacArthur introduced this matter into his most valuable contribution, but I cannot let it pass unnoticed.

The Chairman announced that Mr. Chas. B. Saner's reply to the discussion on his paper, "Notes on a Problem during Shaft Sinking," would be given at the next meeting, and he then asked for discussion on Mr. Cullen's paper, in response to which

Dr. J. Moir (*Vice-President*) read the following contribution :—

THE GASES RESULTING FROM THE USE OF HIGH EXPLOSIVES.

(Read at November Meeting, 1908.)

By Wm. Cullen (Past-President.)

DISCUSSION.

Dr. J. S. Haldane (*Hon. Member*): Having read with great interest the author's paper, I should like, if I am not too late, to contribute a few remarks to the discussion.

He has, I think, completely cleared up the doubts which existed as to the gases produced under ordinary conditions, by the blasting gelatine used in the Rand mines. It is specially interesting to me to find that they are almost identical in composition with those from the high explosives used for similar work in Cornish mines. The 15 experiments given in his paper show a ratio of CO to CO_2, varying from 1 : 6 to 1 : 10, whereas 19 experiments carried out by Messrs. Thomas, Macqueen, and myself (Trans. Inst. of Mining and Metallurgy, vol. xiii., 1904, p. 458), showed a ratio varying from 1 : 5 to 1 : 19, with the gelignite in common use in Cornish mines. The average is nearly the same in the two cases—about 1 : 8 for the Rand mines, and 1 : 10 for the Cornish mines; and even the small quantities of hydrogen formed, agree closely. The author is mistaken in attributing to me the opinion that CO is absent from the products of combustion of the high explosives used in metalliferous mining. It has invariably been present in my experience.

The further investigations, on which Messrs. Cullen and Weiskopf are engaged, as to some means of reducing the proportion of CO formed, are of great importance; and any step forward which they may be able to make in this direction will deserve the gratitude of the mining community.

The author found that as much as 1·28% of CO might be present in the air at the end of a drive directly after blasting, whereas in the Cornish experiments the maximum percentage found was only 0·24. The explanation of this great difference evidently depends upon the differing quantities of explosive used. In the case of the author's observations, about 45 lbs. of explosive were, as a rule, used, for each round of shots, whereas only about 5 lbs. were used in the Cornish mines. Gassing is comparatively frequent in the Rand mines, as has been shown very conclusively by Drs. Irvine and Macaulay, whereas it is so rare as to be almost unknown, in Cornish mines. The need for proper local ventilation by means of air-pipes or other means, to places where development work is proceeding, is thus a very urgent one in the case of the mines of the Rand, and the author's paper places this matter in a perfectly clear and intelligible light.

From some of the author's introductory remarks, I fear that he is inclined to believe that the very high mortality from phthisis among white men engaged in development work, is due more to poisonous gases, than to the inhalation of stone-dust. The Society will, I hope, excuse me if I once more express my conviction that stone-dust, and stone-dust alone, is responsible for the terrible liability to phthisis among these men; and that, important as it is to secure sufficient ventilation to prevent danger or impairment of working capacity from gassing, it is still more important to most stringently carry out the comparatively simple precautions which are needed to prevent inhalation of dust during drilling, after blasting, and during removal of the rock which has been broken.

Some of the most interesting results given by the author are those for the air before blasting. It would seem from these results, that this air often contains as much as about ·03% of CO. Unless these samples were taken soon after "cut" shots had been fired, they indicate a very defective condition of ventilation ; for such air would in the course of an hour or two, seriously affect a man's working capacity. Thanks to Dr. Moir ; much is now known as to the main gaseous impurities in the air of Rand mines ; but more information would seem to be required about the proportion of CO in air where men are actually working. We may safely say that air containing more than about ·015% CO is distinctly objectionable if breathed for long, although about ten times as much would be needed to cause any actual risk to life.

SMALL MINES OF RHODESIA.

(*Read at September Meeting, 1908.*)

By B. I. Collings (Member).

DISCUSSION.

Mr. A. L. Edwards : The author lays great stress on the fact that the miner cannot be too careful in attacking a quartz proposition, that he cannot be over careful, and that he must never take anything for granted, but must always verify his facts, before he ventures to put money into the concern. This dictum, of course, holds good in regard to all mining propositions ; eternal vigilance, the greatest care and foresight, are as necessary in banket, as in quartz ventures. That the author's word of warning is still needed, will be evident from the following two cases which have come to my knowledge.

Syndicate No. 1 started work on some eleven claims leased from Moodie's Exploration Co. The mine had been worked in the early nineties and thrown up, but "other times, other men." A member of the syndicate sampled the mine, the samples were sent to the Rand, and the results obtained were astounding, the lowest assay being 30 dwt., and the highest 1,260 dwt. per ton. The reef varied from 22 in. to 42 in. The imaginary profits were so great as to at once call for the buying and erection of a 5-stamp (750 lbs.) mill, a 25 h.p. gas engine, etc. Crushing operations were commenced in September last, and came to a full stop at the end of December, 1908. Some 1,200 tons of ore, selected and otherwise, were crushed, and gold to the value of £600 recovered—approximately 10s. a ton. The working costs including 7·5% royalty figured at 13s. 10d. a ton, without redemption or sinking fund. From the start to the finish, a sum of £2,000 was expended ; against this outlay, gold to the value of £600 was recovered ; thus each ounce of gold won cost £13, against a selling value of £4·2 an ounce.

Syndicate No. 2.—This syndicate started work on a property which had been abandoned for over twenty years : the syndicate members were told that the mine contained 12,000 tons of 9 dwt. rock, which the former owners had very kindly left behind for a rainy day. A 10-stamp mill was purchased and erected, a 25 h.p. motor purchased, and current arranged for with Moodie's Co. Crushing was commenced, and continued for a period of two months, some 500 tons of rock being crushed, and 20 oz. of fine gold recovered, or 3s. 3d. per ton milled. The working costs on this venture were as follows : European wages 4s., native wages 3s., power, 1s. 6d., stores 2s. 6d., and maintenance, 1s. per ton. The revenue did not balance expenditure ; result — more experience, less money. Every ounce of gold won on this property cost £100, some £2,000 having been sunk in the venture.

Now let us suppose that sense, instead of enthusiasm, had predominated amongst the members of these syndicates, in which case, an unbiassed trained mine sampler would have been employed to sample the mines, the assays made on the Rand, and the results tabulated, and handed to the syndicates for a total outlay of about £100 ; and the syndicates would to-day have been £3,900 in hand. It is one of the most astounding features of some men's egotism, that they conclude, that although they may have been absolute failures in every other walk of life, yet they assume that they, and they only, are the chosen ones to successfully run a gold mine, or work a farm, propositions which call for the very best of trained knowledge.

Mr. Wilfrid Taylor in his contribution to Mr. Collings' paper raises the question, "whether small mines ever do pay?" Well, I think the whole case is fairly stated, and most succinctly summed up, in his concluding statement, namely; "That for every successful venture, there are ten others which never have and never will repay 25% of the money sunk in them.

The meeting then closed.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Method of Separating Metallic Bases (Metals).—"To avoid the inconveniences which arise in the ordinary methods of qualitative analysis from the occasional presence of phosphoric acid, this acid is always added, as sodium phosphate, to the solution under examination. The metallic elements are separated into six groups. Group I., precipitated by hydrochloric acid, contains lead, thorium, mercury, silver and thallium. The filtrate is saturated with hydrogen sulphide and the precipitate separated into Group II., antimony, tin, arsenic, molybdenum, gold, platinum, and iridium sulphides soluble in ammonium sulphide, and Group III., lead, bismuth, copper, cadmium, zinc, mercury, osmium, rhodium, palladium, and ruthenium with insoluble sulphides. To the boiling solution, containing the metals not precipitated by hydrogen sulphide in acid solution, an excess of sodium phosphate, hydroxide, and hypobromite is added. The filtrate contains Group IV., thallium, vanadium, aluminium, glucinum (beryllium), chromium, zinc, potassium, caesium, sodium, rubidium, tantalum and niobium. The precipitate is dried and fused with sodium carbonate, taken up in water, washed and dissolved in hydrochloric acid, boiled to remove carbon dioxide, and Group V., manganese, iron, uranium, indium, gallium, zirconium, cerium, thorium, lanthanum, didymium, titanium, erbium, gadolinium, and germanium, precipitated by ammonia solution. The solution contains Group VI., cobalt, nickel, calcium, strontium, barium, magnesium and lithium. Tables are given for the identification of the metals contained in the various groups, and the author claims that the separations are quantitative and that the method may be used as a basis for the estimation of the various metals." E. E. Pozzi Escot, *Bull. Soc. Chim. Belg.*, 1908, 22, 327—338.—*Journal of the Society of Chemical Industry*, Sept. 30, 1908, p. 962. (A. W.)

Production of Ammonia from Atmospheric Nitrogen by Peat.—"It is stated that a mixture of nitrogen and hydrogen passed over reduced iron at a low red heat produces ammonia. Better yields are obtained by passing a mixture of air and coal gas over iron oxide; oxides of other metals, particularly those of bismuth and chromium, behave similarly. With freshly reduced iron wire gauze, the most favourable temperature is 300—350° C. Further experiments showed the presence of oxygen and water to be of importance, and finally led to the complete omission of the coal gas or hydrogen. As the oxidation of the iron proceeds, the formation of ammonia decreases, and it is necessary to reduce the iron from time to time by passing carbon monoxide or hydrogen over it at a higher temperature. Carbonaceous materials (coke, charcoal, peat) sufficiently cheap to admit of renewal when exhausted, were next substituted for iron, and thus the necessity of periodically reducing the contact material was avoided. Peat was found to give the most satisfactory and rapid results, and even when it contained up to 80% of water, it could be utilised advantageously. To prove the co-operation of atmospheric nitrogen in the change, sugar carbon absolutely free from nitrogen was treated by the process and produced an average of 1% of ammonia on the carbon consumed. In a plant consuming 1 ton of peat in 24 hours, the average yields of ammonium sulphate obtained were, it is stated, 7%. A large plant has been constructed at Carnlough, Ireland." —H. C. Woltereck.—*Journal of the Society for Chemical Industry*, Oct. 15, 1908, pp. 978—979. (A. W.)

The Manufacture of Hydrogen.—"Pure hydrogen manufactured with a simple apparatus and from non-bulky materials, is the achievement of recent French experiments. Fine aluminium filings are mixed with a small proportion of mercuric chloride and potassium cyanide, which causes a slight rise of temperature, and produces a coarse powder, quite stable if kept dry. This powder it treated with water, about 1 litre to a kilogram, and the temperature, as the hydrogen is evolved, is kept below 70° C. by the addition of more water. At that temperature 2¼ lb. are completely oxidised in two hours, and yields 44 cub. ft. of pure hydrogen."—*The Mining and Scientific Press*, Nov. 28, 1908, p. 730. (W. A. C)

Fuller's Earth.—"The term 'fuller's earth' is used to include a variety of substances that possess strong absorbent properties. It was originally used for cleansing cloth of grease, and also for cleaning furs, but its main use in this country at the present time is for the clarifying of oils.

Dana defines fuller's earth as including many kinds of 'unctuous clays, grey to dark green in colour, and being in part kaolin and in part the mineral smectite.' It is placed by him with several clay-like minerals (all of them hydrous silicates), namely, smectite and malthacite, of not very definite composition, but all having a high percentage of combined water. It is inferred that Dana considers these minerals the cause of the bleaching power of fuller's earth.

Ries defines fuller's earth as a clay-like substance that has the property of decolourising or clarifying oils. He says:—'An ultimate chemical analysis shows it to differ from most ordinary clays in having usually a high percentage of combined water and a low amount of aluminium. There is probably a large amount of hydrous silica present. Fuller's earth possesses little or no plasticity, and in order to work properly has to be ground very fine. A chemical analysis is of little value at present in determining its quality; only a practical test suffices.'

Geikie defines fuller's earth as 'a greenish or brownish, earthy, soft, somewhat uncuous substance, with a shining streak, which does not become plastic with water, but crumbles down into mud. It is a hydrous aluminous silicate with some magnesia, iron oxide, and soda.' According to Geikie, fuller's earth owes its detergent properties to physical characters rather than to chemical composition.

John T. Porter, in an article on the 'Properties and Tests of Fuller's Earth,' presents a very interesting theory. He says that it is evident that Dana's theory will not stand since the discovery of American deposits having a comparatively low percentage of combined water. Such earths could not have as their base either smectite of malthacite, although they might contain very small quantities of these minerals.

Concerning Ries's statement that fuller's earth probably contains much hydrous silica, inferring that it is this constituent which gives it its distinctive qualities, Porter says:—

'My analyses have confirmed Ries's statement that fuller's earth contains appreciable amounts of hydrous silica. However, it is evident that if the bleaching power of fuller's earth is due to this hydrous silica; treatment of the earth with boiling carbonate or hydrate of soda, which removes the silica, should destroy this bleaching power. I have tried this experiment on several earths, and although the results are not entirely concordant, yet there can be no doubt that fuller's earth retains at least a part of its efficiency after treatment by alkalies. It is also plain that carbonate of soda has a much less harmful action on the earth than the hydrate. As Fresenius and other authorities state that hydrous and amorphous silica are freely soluble in hot carbonate of soda, as well as in sodium hydrate, this difference can hardly be due to non-solution of the silica by the carbonate, but is almost entirely due to partial decomposition of the earth by sodium hydrate. In this connection I have noticed that sodium hydrate appears to extract considerable alumina as well as silica, but the carbonate does not. Fuller's earth after treatment with sodium hydrate is left in a very gelatinous condition, and is extremely hard to filter and wash. For this reason it is possible that some of the samples treated retained considerable amounts of soluble salts after washing, although great pains were taken to remove them, and this may have had some influence on the results.

'Another fact which might be used as an argument against this theory is that hydrous silica artificially prepared has but very slight bleaching powers. These results are naturally not conclusive, as the silicic acid occurring naturally may differ either physically or chemically, or both, from the artificial product.

'On the other hand, some earths still retain a considerable portion of their bleaching power after decomposition by acids, consisting then of 80% or more of silica: and from this it seems probable that hydrous silica when prepared in certain ways may have some small efficiency.

'In conclusion, I think I am justified in stating, first, that hydrous silica does not of itself possess bleaching power, although it may at times possess some efficiency as a result of existing in a certain physical condition; and, second, that it is certain that fuller's earth cannot owe more than a small part of its peculiar properties to the presence of free silicic acid.'

Porter offers a new theory to explain the clarifying action of fuller's earth. It is based on the presence of a group of aluminium hydro-silicates existing in the form of pectoids. He says:—

'The theory which I have devised to explain the peculiar properties of fuller's earth may be stated as follows:

'1. Fuller's earth has for its basis a series of hydrous aluminium silicates.

'2. These silicates differ in chemical composition.

'3. They are, however, similar in that they all possess an amorphous colloidal structure.

'4. The colloidal structure is of a rather persistent form and is not lost on drying at a temperature of 130° C., or possibly higher.

'5. These colloidal silicates possess the power of absorbing and retaining organic colouring matter, thus bleaching oils and fats.

'I have used the word colloidal in this statement in its broadest sense—to cover the whole range of conditions expressed by the words colloid, pectoid, and hydrogel. It is my opinion that the word pectoid would most properly express the condition of the active constituents of fuller's earth, but it is not impossible that these may go into partial solution in oil and thus become true colloids.

'Colloids are known to possess the power of taking up organic colours from solution, and the analogy with fuller's earth is so apparent as to excite surprise that it has not been studied from this point of view before. It has even been known that pectoids could extract colours from oils and resins as well as aqueous solutions. This I regard as a strong point in favour of the colloidal theory, although by itself it does not, of course, constitute a proof.

'The power of colloids to absorb certain salts, or at least the basic ions of these salts, has been known a long time. It is also possessed to a degree by certain amorphous substances having a fine porous structure, such as charcoal and boneblack. Fuller's earth has this property to a marked degree. In fact, its use has been proposed, to remove the lime from boiler water. I have found that after it has absorbed ammonia or salts, it loses a great part of its efficiency in bleaching oils. From this I infer that the bases are absorbed in a similar manner to the colouring matter of oil, and occupy the pores which otherwise would hold the colour.

'It has been pointed out that these absorbent materials have a selective action on the salts, absorbing the unlike ions and discarding the ions of like chemical nature. If colours in oil solution are absorbed in the same manner as salts in aqueous solution, analogy would lead to the supposition that fuller's earth would exert a selective action for certain classes of colouring matter, and moreover, that the earths themselves would differ in their selection of colours according as they are more or less acidic in composition. We find this entirely in accordance with facts.

'I have noticed in the course of my oil tests, that on the same sample of oil different materials give products varying greatly in shade, the main colour being in some tests of a yellow and in others of a green tint. Is it not likely that the oil contains several compounds of varying chemical nature, and that the earths or other materials used in bleaching, extract them in ratios proportional to their own basicity or acidity. I have been unable to obtain any exact data on this point from my results, owing to the difficulty of following these slight changes in tint with the unaided eye. A tintometer would be needed if this line of investigation were to be followed out.'

Mining and Preparation.—The fuller's earth in Florida is usually mined by pick and shovel, and hauled to the mill, where the earth is broken up into small lumps and put through rotary driers. It is then taken by elevators to the crushers and ground and bolted into several grades, 120 mesh being the finest. The material is then ready for shipment."—F. B. VAN HORN, *Mineral Resources of the United States.—Mining Science*, Dec. 3, 1908, pp. 450—451. (A. R.)

PERMANGANATE METHOD FOR DETERMINING COPPER.—"*Details of the Method.*—Weigh 0·5 gm. of ore, or more if very low in copper, into a tall 250 c.c. beaker and add 10 to 12 c.c. of a mixture of sulphuric and nitric acids (1 : 3); if the ore contains talc, clay or similar minerals making filtering slow, add 4 to 8 drops of hydrofluoric acid, place on a hot plate and boil rapidly to dense fumes of sulphuric anhydride Remove from the hot plate and cool slightly. Add 35 c.c. of water and neutralise with ammonia, avoiding excess. Add 4 c.c. of hydrochloric acid; follow this with 10 c.c. of a solution of sodium sulphite (200 c.c. per litre) to reduce the copper and iron salts. Cover with a watch glass and set on the hot plate until the solution begins to boil; when 5 c.c. of a solution of potassium thiocyanate (40 gm. per litre) are added to precipitate the copper. Boil for one or two minutes, but not long enough to expel all sulphurous acid. Remove from the hot plate and, with the beaker still covered, allow to settle for about five minutes. Filter and wash thoroughly with a minimum amount of hot, but not boiling, water. Replace the original beaker under the funnels and wash the copper thiocyanate with a boiling hot 6% solution of sodium hydroxide, which decomposes the cuprous thiocyanate. The sodium hydroxide is best added from a wash bottle, and the jet directed on the precipitate in such a manner that it is thoroughly stirred, insuring complete decomposition. The cuprous thiocyanate and sodium hydroxide react according to the following equation:

$$CuCNS + NaOH = CuOH + NaCNS.$$

The soluble sodium thiocyanate goes in the filtrate, while cuprous hydroxide being insoluble, remains on the filter. After washing thoroughly, acidulate the filtrate with sulphuric acid, and titrate the liberated thiocyanic acid with permanganate.

Corrections Necessary.—Unfortunately, copper thiocyanate is not absolutely insoluble, and allowance must be made accordingly. Under the conditions given above a nearly constant amount of 0·005 gm. copper, equivalent to 0·1% on 0·5 gm. assay, remains in solution, lowering the reading by that amount. A partial compensation for this error is made by the small amounts of oxidisable matter in the sodium hydroxide and from the filter paper which raises the result 0·05 c.c. In summing up these two errors, the reading will be only 0·05% too low on 0·5 gm. assay.

The strength of the permanganate solution is such that 1 c.c. is equivalent to 0·01 gm. of iron. If the iron value of the solution be multiplied by 0·197 and 0·0025 gm. added (for the copper left in solution) the result is equal to the copper content. Thus on 0·5 gm. assay, 1 c.c. of permanganate is equivalent to nearly 0·04% copper, or 0·02% on 1 gm. assay. A table may easily be made to convert the readings for iron to percentages of copper.

Where extreme accuracy is not required, the permanganate solution may be so made that each 1 c.c. equals 1% copper on 0·5 gm.; then, in standardising and in regular work add one drop, or about 0·05 c.c. excess to allow for the copper left in solution. This solution requires about 14·5 gm. potassium permanganate per litre. Much time is saved by making all solutions of definite strength and keeping them in bulk. These solutions are added to the assays from large 500 c.c. dispensing burettes. With a little practice one can judge the amount used so closely that the reading need not be taken.

Precautions.—If the amount of hydrofluoric acid used does not exceed a few drops it will not sensibly affect the beaker or large particles of ore, but will effectually decompose the slimes. I have used it 25 times or more in the same beaker before the glass showed signs of etching. This treatment with hydrofluoric acid lessens the tendency to spit and bump, while boiling off the nitric acid and leaves the assay in a condition to filter rapidly later on. Under proper conditions, hydrofluoric acid can also be used to advantage in other determinations that are to be made on ores difficult to filter; it is useful in the iodide method for copper, and the molybdate method for lead.

If the solution turns red (and stays so) after adding the thiocyanate, it indicates that some iron is in the ferric state, and that more sodium sulphite should be added to reduce it. Ferric salts react with cuprous thiocyanate, oxidising it, and the results are then slightly low.

Five cubic centimetres of potassium thiocyanate solution will precipitate 0·125 gm. of copper; therefore, if the ore contains more than that amount, more thiocyanate should be used, but a large excess is to be avoided.

Hydrochloric acid may be present up to 4%, but above that amount it noticeably increases the solubility of the cuprous thiocyanate. Free sulphuric acid should not be present except in very small amounts.

When titrating, the solution should have a bulk of at least 100 c.c.; otherwise some of the hydrocyanic acid formed in oxidising the thiocyanic acid may escape into the room with injurious effect.

When extreme accuracy is desired it is best to measure the reagents, as the results are somewhat affected by any considerable variation in the quantity used. In the permanganate method, as in most others, some experience and careful attention to every detail is necessary to secure the best results."—F. G. Hawley.—*Engineering and Mining Journal*, December 12, 1908, p. 1,155. (A. R.)

METALLURGY.

Continuous Slimes Solutions Filter.—"The method followed in the construction of these filter bottoms for continuous slime solution filtering at the Black Oak mine was as follows: upon the floor of the vat were first laid pieces of 2 × 2 in. scantling, planed, about 12 in. apart, and parallel to one another. These were notched at intervals on the under side to allow free circulation of the solution. Resting on these scantlings, triangular pieces (made by cutting 2 × 3 in. scantling diagonally) were placed. The edges of the bases were placed about ½ in. apart, the ends conforming to the circumference of the vat. In the V shaped spaces was first laid sharp quartz passing a ¾ in. screen. Enough of this material was used to act as a retaining floor for the next layer of material, which was ¼ in. clean quartz. This was used to fill the V spaces to about 1 in. from the top. Sufficient coarse sand (10 mesh) was then put in to level off. It would be natural to suppose that the sand and slime would eventually work down to the apex of the V spaces, but it does not seem to do so to any appreciable extent. As soon as the filter has bedded itself, there will be no trouble with muddy solution. I have frequently used this type of filter in laboratory work, where I have found it equally efficient."—E. N. Walker.—*Mining and Scientific Press*, Oct. 17, 1908, p. 524. (K. L. G.)

Home-Made Cyanide Plant.—"The following description of a home-made cyanide plant may prove interesting in showing how low grade tailing, formerly thrown away, is being treated at a profit by a couple of men, at small cost of labour and supplies, and a trifling initial expenditure of capital. This little plant treats the tailing from the 40 stamp mill of one of the large mines of the Grass Valley district, California. The management of the mine has not felt justified in going to the expense of erecting a large cyanide plant, as the tailing assays only from 40 cents to $1 per ton. As the men who put up the plant had no contract with the mine for the tailing, they naturally sank the least possible amount of money in the enterprise, in view of the precarious source of their supply of material.

The mill tailing is collected below the mill in sluice-boxes, and carried a half mile down the ravine in V shaped overhead launders. It is then allowed to flow directly. without previous sizing, into one of a series of five leaching vats. These vats are each 18 ft. diameter and 5 ft. 6 in. deep, constructed of 2-in. redwood, and hold 40 tons of sand. One charge is obtained in 24 hours. The distributor for the sand was constructed on the ground, and has some special features, having the merit of so mixing the sand and slime that a charge with as much as 20% slime can be successfully leached. It has six arms, 7 ft. long and 20 in. wide; each of these is divided into 12 separate channels, down which the sand flows from a round central sand-box. Each channel comes to a stop before the end of the arm is reached, and a hole is bored through the bottom, through which the sand drops into the vat. These holes are so arranged that the sand is evenly distributed. Hence, there is no need of levelling the vat when it is filled. The surface is almost as level as a billiard table, and solution can be run on at once. Before entering the channels, the sand flows into a round box which is pierced at the bottom with 1-in. holes, through which the pulp spurts evenly. The launder that feeds the sand box has four large holes, insuring even distribution without packing. Unlike most distributors, this one is lowered into the vat to be filled, and raised gradually by a windlass, a barrel of rocks serving as a counterbalance. A large proportion of the slime is floated off immediately through a system of plug holes, of which there are three rows, 6 in. apart, the holes being 2 in. diameter. As fast as the vat fills, a plug is put in and the one next above it is knocked out; thus there is no loss of sand, but the slime has no chance to settle, as might be the case if the holes were only at the top of the vat.

Instead of using the familar lawn-sprinkling device, the distributor is turned mechanically by a system of old cogwheels and bicycle chains at about one revolution every 7 minutes, which is slower than most distributors. The initial power is furnished by a small stream of water falling on a home-made overshot water-wheel. The distributor travels back and forth over the leaching vats on overhead tracks, and is pulled along by a windlass and wire ropes fastened at either end. The vats are emptied by 'hosing out' the sand through a sluice-gate at the bottom, 6 × 6 in. square. This takes about 3 hours.

The sand is leached in the regular manner, and receives a 96-hour treatment. Two strengths of solution are used, the first of about 0·2%; the second is weaker: about 10 tons are introduced at one pumping, about 100 to 125 tons of solution being used altogether. No lime is added. Between each batch of solution the charge is drained for 1 to 5 hours, depending on how the sand leaches. The cyanide consumption amounts to about 0·6 lb. per ton, and the extraction is generally above 85%.

The zinc-boxes are half barrels, caulked and painted. The barren solution flows into a sump, from which it is pumped direct, after the necessary cyanide has been added, to the leaching vats, by a small Dixon pump, which, being valveless, requires little attention. A 12-ft. wooden overshot water-wheel, constructed on the ground, gives ample power. The precipitated gold is dried on an open hearth-furnace, the zinc being partly volatilised. It is then sent to a local assayer for refining, and sold to the mint."—W. F. BOERICKE and B. L. EASTMAN.—*Mining and Scientific Press*, Nov. 21, 1908, p. 712. (A. R.)

BATTERY AND CYANIDE COSTS.—"Many and difficult are the problems to be deduced from the question asked by a correspondent as to the actual cost of battery treatment of gold ores, and their cyanidation, in Australasia. Comparatively few mines, to their discredit, keep these items distinct, and even as set out in their financial statements their elucidation would puzzle the highest accountant. Bookkeeping unfortunately does not form a special feature of the work of most mining managers, and access to the books would not aid one in attempting to give an accurate statement of costs. Then there are the varied conditions which obtain in different mining centres. The ore may be 'dirty,' or 'even when the materials are hard, the toughness, specific gravity, mesh aperture, and depth of discharge and quantity of water used have an influence which greatly modifies the output.' 'Mills,' says Donald Clark, in 'Australian Mining and Metallurgy,' published by this journal, 'of the same type running on stone from Ballarat, and the heavily mineralised ore at Charters Towers, will give a widely different output. Where 20% of heavy metallic minerals is present, a battery will seldom crush more than 60% of the tonnage it would put through in the same time if the material were clean quartz.' Still by dissecting several of the financial statements of mines in Western Australia, which, by the way, devote more attention to the preparation of cost sheets than mines in any of the other States of the Commonwealth, it is possible to afford some information. The figures of several of the mines on the Witwatersrand, where free milling and cyanidation obtain on a large scale, have also been prepared. The following particulars have been gleaned:—

Western Australia—

Golden Ridge, milling, 2s. 7·61d. per ton.

Ivanhoe, milling, 2s. 11·29d.

Associated Gold Mines, milling, 1s. 11·15d.; agitating and cyaniding, 1s. 6·35d.

Great Boulder Proprietary, cyanide mill expenses, 3s. 1d. per ton (2,000 lb.), sulphide mill expenses, 7s. 6d.

Golden Pole, milling, 5s. 11·14d.; cyaniding sands, 4s. 4·51d.; cyaniding concentrates, £1 1s. 3·71d. per ton.

Sons of Gwalia, milling, 4s. 7·19d; cyaniding sands, 2s. 0·78d.

Lake View Consols, milling, 1s. 8·07d.; cyaniding concentrates, 4·74d.; cyaniding by agitation, 2s. 8·69d.

Rand—

Witwatersrand G. M. Co., milling, 1·541s.; cyaniding, 1·632s. per ton.

New Rietfontein Estate, milling, 1·694s.; cyaniding, 2·632s.

New Unified, milling, 2·345s.; cyaniding, 1·549s.

Consolidated Langlaagte, milling, 2·454s.; cyaniding, 1·259s.

New Goch, milling, 1s. 10·085d.; cyaniding, 2s. 0·61d.

Van Ryn, milling, 1s. 9·503d.; cyaniding, 1s. 9·067d.

Meyer and Charlton, milling, 2s. 5·469d; cyaniding, 2s. 2·745d.

Roodepoort United Main Reef, milling, 2s. 0·842d.; cyaniding, 2s. 9·218d." — *The Australian Mining Standard*, Dec. 9, 1908, p. 654. (A. R.)

METALLOGRAPHIC STUDY OF ORE DEPOSITS.—"A few examples of the application of the methods of microscopic metallography to the study of ore deposits, taken from the concluding section of Dr. Campbell's paper are given below.

'The latest development of metallography is its application to economic geology. By its aid we can distinguish the relative ages of the various opaque constituents of ore bodies much more easily than can be done by hand specimens or in the petrographic slide, when dealing with complex and compact masses.

'The ordinary specimens from Butte, Mont., are composed of iron pyrites with more or less copper. Under the microscope the pyrite is clearly the oldest constituent. It has been broken and fractured, and then eroded by solutions. Then in the interstitial spaces were deposited bornite and chalcocite. The chalcocite is apparently younger than the bornite, for it cuts it in places. Very often when the specimen shows chalcopyrite this latter was the last to form, because it is the groundmass of the pyrite, bornite and chalcocite.

'The silver deposits of Cobalt, Ont., have been studied in this way. We find that the first mineral to crystallise out in the vein was smaltite, and this was followed by niccolite, for cubes of smaltite are found embedded in niccolite. Both the niccolite and the smaltite show signs of disturbance and are cut by veins of calcite. In a section from the La Rose mine rough smaltite is seen enclosed in smooth-polishing niccolite, both of which are cut by thin veins of calcite, which appears black on account of the vertical illumination. Of later age still is the argentite, which cuts the calcite: while the silver cuts both argentite and calcite. The bismuth came down with or a little later than the native silver. Thus we can establish the order: smaltite, niccolite, period of disturbance, calcite, then argentite, native silver and bismuth. In addition, we find crystals of cobaltite incrusted on the rosettes of smaltite (cloanthite) embedded in the calcite, therefore the cobaltite is slightly younger than the smaltite and older than the calcite. Mispickel occurs like cobaltite. It is well known that much of the silver in the Cobalt deposits is not pure. This is explained when it occurs as veins, for each vein has a thin envelope of a bluish harder substance which polishes somewhat in relief, probably a native alloy of silver.

'Nickeliferous pyrrhotites have long been the subject of discussion. Many hold that the nickel replaces the iron isomorphously. We have examined specimens from widely different localities, and in each case the nickel occurred as pentlandite. Chalcopyrite usually occurs also, and we find the following order of succession holds good: pyrrhotite, pentlandite, chalcopyrite. Secondly, their origin is much discussed. Are the deposits of direct igneous origin or have they been deposited through the agency of solutions? The specimens we have examined show such a structure that they could not have separated from an igneous mass. They show no resemblance to nickel mattes.

'The process of decomposition and of secondary enrichment can be studied metallographically. . . Another important line of work is the study of certain complex mineral species to determine their constitution. We can ascertain in many cases whether a mineral owes its peculiarity of formula to a definite combination or to the presence of foreign bodies as in the case of a mechanical mixture. In the majority of specimens examined, there is found more or less admixture of foreign matter. Chalcopyrite includes chalcocite or pyrite, sometimes even galena. Tetrahedrite includes quite a number of other minerals, and so on. Steel galena when examined is found in many cases to owe its fine structure to the presence of a second mineral. Each grain is surrounded by a fine film of quartz in one case, calcite in another, tetrahedrite in another, blende in another, and so on. In many cases the galena was deposited, then crushed and the second constituent then deposited. Minerals often show the effects of strain when etched, especially galena and pyrrhotite.'"—DR. W. CAMPBELL. — *Engineering Magazine*, Dec., 1908 (J. A. W.)

MINING.

EXPLOSIONS IN MINES—EXPERIMENTAL STATION. —"M. Barthou, the Minister of Public Works, inaugurated the new experimental station in connection with the Liévin mines, when some interesting demonstrations took place in his presence. This testing establishment, now the most richly endowed institution of the kind in Europe, was founded immediately after the disaster at Courrières. This catastrophe, which caused the loss of upwards of 1,000 lives, revealed the existence of a grave and hitherto unsuspected source of danger, arising from the presence of coal dust. The powerful ventilation, necessary in the case of fiery mines, to ensure the aeration and the safety of the workings, by the removal of the fire-damp, gives rise at the same time to dust, which may, under certain conditions, cause formidable explosions.

Equipment of Laboratory.—In order to carry out a systematic and scientific investigation into the circumstances which may tend to favour or to hinder such explosions, the creation of this laboratory was decided upon. The Central Committee of French Collieries voted a sum of £12,000 for the cost of providing and equipping the institution, together with a further contribution of £3,000 a year for maintenance. M. Taffanel, a mining engineer, was selected by the Government to take charge of the experiments. After visiting the Gelsenkirchen laboratory in Westphalia, and that of Frameries in Belgium, he prepared designs for the laboratory at Liévin, the works in connection with which were completed in July last. This establishment is situated in close proximity to an electric generating station, and to a mine which is of a somewhat fiery character. The station comprises a central block of buildings, with offices, library, laboratories, and two gasometers, one for the storage of fire-damp escaping from the mine, where the gas is kept ready for use when required for experimental purposes, and the other for atmospheric air. This latter is connected with a turbine pump, electrically driven, capable of supplying air at an extra pressure of one-eighth atmosphere, which can be utilised for the preparation of explosive mixtures.

A coal storage shed is annexed to the main building, divided into compartments, each of which is reserved for a special description of coal undergoing tests. A complete plant for crushing and grinding coal enables specimens of dust of any required degree of fineness to be prepared, and this dust can be conveyed thence to the testing gallery by means of conduits in communication with a fan.

Gallery for Tests.—The testing gallery consists of the main explosion chamber, the imperviousness of which is assured by means of partitions or diaphragms formed of waxed packing paper. In continuation of this first chamber there is a gallery, constructed of reinforced concrete, the reinforcement of which consists of steel bars 30 mm. in diameter. Beyond this there is a further gallery, timbered in the same way as a colliery working. The length of this at the present time is 75 metres, but it will ultimately be extended to 500 metres, and lateral workings will be added in such way as to reproduce as nearly as may be possible the conditions existing in an actual coal-pit.

Observation Chamber.—An observation chamber, well screened or guarded, annexed to the testing galleries, will enable a distant view to be safely obtained of the effects of the explosions. Two very massive erections of reinforced concrete, placed at the mouth of the gallery, serve to keep in place the heavy timbers used to close the entrance during the tests. In order to cause an explosion, recourse is had to two powerfully made cannons, which can be fired by an electric current set in action from the observation chamber.

Practical Working Tests.—1. In an atmosphere containing 450 gm. of dust per cubic metre, 320 gm. of the explosive known as 'grisoutine-couche' (fire-damp allayer) were fired without causing the dust to become ignited. Previous tests made with still larger quantities of this explosive had led to similar results. It may thus be laid down that the use of safety explosives secures absolute immunity from risk.

2. The same experiment conducted with 80 gm. of dynamite jelly caused a violent explosion, by which flames and smoke were ejected beyond the testing gallery. The speed of flame propagation reached 80 metres per second.

3. The identical experiment was repeated, only the dynamite was caused to act on a mixture of atmospheric air containing 10% of fire-damp. The explosion in this case was still more violent. The fire-damp does not, however, add to the length of the flame, but increases the rapidity of its transmission.

4. Trials were made in a similar way with a dust-laden atmosphere containing either 40% of ashes or of a powdered incombustible schist. The after-explosive action no longer takes place with dust of this character, even if dynamite is used. This is a very interesting result, as most of the mines have air highly charged with dust of this nature, and in all cases it is easy to bring about this state of things artificially. This would be a safer proceeding than the suggested system of watering in the case of a large number of mines."—*Times Engineering Supplement*, Nov. 25, 1908. (J. A. W.)

ELECTRIC COAL WINDING IN SOUTH WALES.—"At a special general meeting of the South Wales Institute of Engineers, held at Cardiff on December 1, Mr. Gerald Hooghwinkel read a paper on 'Electric Winding at the Maritime Pit of the Great Western Colliery Company.' The author said that the electric winder installed at this colliery was the first electric winder of moderate capacity at work in England and Wales, and in the South Wales coalfield the two chief points which in recent years had altered the conditions in favour of electric winding were the increasing value of small coal and the decreased cost of electric power, the apparent anomaly between these two statements being due to the improved economy in the boiler house and the introduction of large and economical units. In this case power was supplied to the winder by the Consumers Company of the South Wales Power Company at a rate which worked out at slightly over 0·5d. per unit. It was required to wind 175 tons of coal per hour from a depth of 1,110 ft. in 70 winds, giving 5,600 lb. of coal per wind in two trams. The general relations between load and depth were such that no very heavy masses had to be accelerated, and the point at issue was whether the Ilgner equaliser plant of the Westinghouse system should be adopted. Both systems had their advantages, but the main point which decided the management to adopt the Westinghouse system was that in this system a three-phase motor was used capable of being run direct from the mains. In the event of a breakdown of the equaliser set a stoppage at the pit was thereby avoided, and winding could proceed at a reduced output. In winding men this point had also to be considered. The absence of recuperative power with this system was met by an ingeniously designed drum which, if properly handled, stopped without any breaking being required after current was cut off. In the Ilgner system the total fluctuating power demanded by the winding engine passed through the equaliser set was in series with the winding motor. A smaller, and therefore cheaper, equaliser arrangement should be arrived at, if this equaliser set could be run in parallel with the main motor, only supplying and absorbing the amount of surplus power required above and below the average in the same way as a battery of accumulators with a reversible booster floating across the mains in a tramway station. Having given a detailed description of the plant, and directed attention to its principal points, Mr. Hooghwinkel said the plant had been in continual use for about four months, and, although the quantity of coal available from the working was only small, the engine had proved to be easily able to deal with the requirements. A test made on November 18 showed that the cost of winding was well under 2·23d. per ton. Ordinary conditions prevailed during the trial, and it was noticed that the banking could be performed in something like eight seconds."—G. HOOGHWINKEL.—*Times Engineering Supplement*, Dec. 9, 1908. (J. A. W.)

COLLIERY EXPLOSIONS.—"Although many explosions of fire-damp have been ascribed to ignition of the gas by sparks from steel tools in contact with hard stones, it has always been difficult to verify this hypothesis. Moreover, laboratory experiments seem to demonstrate that pure fire-damp never ignites by contact with such sparks, and that explosion is only possible when this gas is mixed with pure hydrogen or heavy hydrogen carbides, which very rarely occurs in mines. Mr. Volf, however, gives an account of an accident in the Beaulieusart Pit, Belgium, ascribed to sparks produced by a pick striking against a spherosiderite nodule. He demonstrates that, conformably with the results of laboratory experiments, the fire-damp contained free hydrogen from the beds between the coal seams. This hydrogen may also be produced by decomposition of water in contact with incandescent coal during fires, and it is by mixture of the gases disengaged with fire-damp that he explains the great inflammability of the latter near burning places. The author also studies the influence of different other gases on ignition and combustion of fire-damp."—*Génie Civil*, Paris.—*The New Zealand Mines Record*, Dec. 16, 1908, p. 215. (A. R.)

COLLIERY RESCUE CHAMBERS.—"Austrian mines are provided with underground rescue chambers. They are made to hold fifty men, being used as retreats in cases of danger. Each chamber is provided with all kinds of food, also water, first-aid supplies, and individual rescue apparatus of the most modern type. Pure oxygen is laid down in pipes from the surface to the chambers. Storage battery lamps are kept in these chambers, and at intervals of 500 yd. along main roads."—*The New Zealand Mines Record*, Dec. 16, 1908, p. 215. (A. R.)

PREVENTION OF MINE EXPLOSIONS. — "The United States Geological Survey issued, under date of October 22, a report on the prevention of mine explosions, submitted by three foreign experts, Victor Watteyne, Inspector General of Mines, Belgium: Carl Meissner, Councillor for Mines, Germany; and Arthur Desborough, H.M. Inspector of Explosives, England. These engineers were in the United States for six weeks at the invitation of the United States Government to assist the Federal authorities in beginning the investigations authorised at the last session of Congress.

RECOMMENDATIONS.

A.—Selecting the Explosives to be Used.—1. We recommend that the Government of the United States examine the explosives now and hereafter used in mining, with a view to eliminating the more dangerous explosives, and to improving and standardising such explosives as may be considered most suitable for such use, these to be designated by the Government 'permissible explosives.'

The term 'permissible explosives' is suggested for the reason that no explosives are entirely safe, and all of them develop flame when ignited; and we advise therefore against the use in the United States of the terms 'safety explosives' or 'flameless explosives,' as these terms may be misunderstood and this misunderstanding may endanger life.

2.—We recommend that the operators and miners of coal use only such explosives as are included in a list of 'permissible explosives,' when the same has been published by the Government, in all mines where there is risk of igniting either dust or gas, selecting that one which their own experience indicates can be used to the best advantage under local conditions.

3.—We also recommend that investigations be conducted to determine the amount of charge of such 'permissible explosives' which may be used to the best advantage under different conditions, with a view to reducing danger to the minimum.

B.—Carrying the Explosives into Mines.—1.—All explosives should be made into cartridges and placed in closed receptacles before being carried into the mine, and the quantity carried into the mine during one day by any miner should be limited as nearly as practicable to the quantity needed by him for use during that day. Handling loose explosives and making them into cartridges by an open light in the mine should be prevented.

2.—Detonators or caps should be handled with great care, and should be carried only by a limited number of responsible persons.

C.—Use of Explosives in the Mine.—1.—Shooting in or off the solid should not be practised.

2.—The depth of the shot-hole should be less by at least 6 in. than the depth of the cutting or mining. The use of very deep shot-holes should be avoided as unnecessarily dangerous.

3.—The overcharging of shots (the use of a larger charge than is required to do the work satisfactorily) should also be avoided as unnecessary and dangerous. The proper standardisation of explosives used in coal mining will greatly facilitate the carrying out of this recommendation. (See also 'A,' 1.)

4.—Shots should never be tamped with fine coal or material, containing coal. Clay or other suitable material should be supplied and used for this purpose.

5.—The firing of two or more shots in one working place except simultaneously by electricity, should not be allowed until a sufficient interval has elapsed between the firings to permit an examination of the working place, in order to see whether any cause of danger has arisen.

6.—Before a shot is fired the fine coal should be removed from the working place, as far as practicable, and the coal dust on the floor, sides, and roof, for a distance of at least 20 yd. from the place where the shot is to be fired, should be thoroughly wet, unless it has been demonstrated that the dust in the mine is not inflammable. (See also 'E,' 1.)

7.—If gas is known to occur in the mine, no shot should be fired until, in addition to the watering, an examination made immediately preceding the time for firing, by a competent person, using a lamp which will easily detect 2% of gas, has shown the absence of that amount of gas from all spaces within 20 yd. of the point where the shot is to be fired.

8.—Believing that such will be one of the greatest advances which can be made in safeguarding the lives of the miners, we recommend the adoption of a system of electric shot-firing, in all mines where practicable, by which all shots in the mine, or in each ventilation district of the mine, may be fired simultaneously, at a time when all miners and other employés are out of the mine.

D.—Keeping the Mine Roadways Clean.—1.—The roadways of the mines should be kept as free as possible from loose coal which may be ground into dust and of rubbish in which such dust may accumulate, in order to facilitate the removal and wetting of the dust.

E.—Wetting the Coal Dust.—1.—In all coal mines where explosives are used it is desirable, and in all mines containing gas it is highly important, that the dust on the walls, timbers, and floors of the working places and roadways should be kept continually wet prior to and during the work in the mine. If, however, conditions of roof or lack of water render this general watering impracticable, at least the dust within 20 yd. of each shot should be wet before each firing, and other precautions against explosions should be practised with unusual care.

It is our opinion that a system of watering which occasionally sprinkles the floor only and leaves dry the dust on the walls and timbers of the roadways is useless, and is also dangerous in that it may generate an unwarranted feeling of security against an explosion.

F.—Special Precautions for Mines Containing Gas.—1.—In any mine where as much as 2% of gas can be detected by suitable methods only locked safety lamps of an approved type should be used so long as such condition exists or is likely to recur.

All safety lamps should be maintained in good condition, cleaned, filled, kept in a special room at the surface, and carefully examined, both when delivered to the miner and when returned by him at the close of each day's work. A defective safety lamp is especially dangerous because of the false feeling of security it engenders.

In the filling of lamps with benzine or other low flash oils, which should always be done at the surface, special precautions against fire or explosions should be taken.

G.—Use of Electricity.—1.—Electricity in mining operations offers so many advantages, and has been so generally adopted, that no reasonable objection can be made to its use under proper restrictions. The electrical equipment, however, should be installed, maintained, and operated with great care, and so safeguarded as to minimise danger from fire or shock. The fact that the effectiveness of some insulating materials is soon destroyed in most mines should not be lost sight of.

We recommend the following precautions: For distribution underground the voltage should not exceed 650 direct current or 500 alternating current, these voltages being intended for transmission to machinery operating at 500 volts direct current and 440 volts alternating current, respectively. Even lower voltages are preferable. The trolley wires should be installed in such manner as to render shocks least likely; that is, placed either high enough to be beyond easy reach or at one side of the track and properly protected.

Where current at a potential of more than 650 volts is employed for transmission underground, it should be transmitted by means of a completely insulated cable; and where a lead or armoured covering is used, such covering should be grounded.

In all mines having electric installation special precautions should be taken against the setting on fire of coal or timber. Enclosed fuses or cut-outs are recommended, and each branch heading should be so arranged that the current may be cut off when necessary.

No live electric wire should be permitted in that part of any mine in which gas is found to the amount of 2%.

In all mines producing gas in dangerous quantities, as indicated by a safety lamp which will detect 2% of gas, the working places should be examined for gas by a qualified man using such a lamp immediately before any electric machine is taken or operated there.

H.—Precautions against Miscellaneous Accidents.—1.—In all new construction, shaft lining, and superstructures about the entrance of the shaft (or slopes or drifts) should be built as far as practicable of noncombustible materials.

About the entrances to mines every possible precaution should be taken to prevent fires or the injury of the equipment for ventilation and haulage. Ventilating fans should be placed at one side of the mine opening, and hinged doors or light timbering should render easy the escape of the explosive force in direct line of the shaft or slope.

Proper precautions should be taken for immediately preventing the entrance into the mine of heat and gases and for facilitating the escape of the men in case of surface or shaft fires.

2.—The surface equipment for handling the coal should be so arranged as to prevent coal dust from entering the mine shaft.

3.—In all new mines, and in all old mines, so far as practicable, suitable man roads should be provided for the men, separate from the main haulage roads.

4.—In connection with the system of ventilation it is recommended that in the more frequented roads connecting the intake with the return air-courses, two doors be provided, these doors to be placed at such a distance apart that, while one is open, the other is closed.

5.—In view of the large number of accidents from falls of coal or roof, under the existing practice with single props, more attention should be given to the introduction in mines where the roof is bad of better systems of timbering, such as have been long in use with economy and safety in many well-managed mines.

6.—In undercutting coal by hand, the premature fall of the coal should be prevented by sprags or other suitable support.

7.—We believe that the difficulties and dangers encountered in the working of coal seams which are thick and steeply pitching, or of which the coal is highly inflammable in character or subject to firing from spontaneous combustion, and mines where the subsidence of the surface must be avoided, may be successfully and economically overcome in many cases through the adoption of the flushing system of mining—that is, the filling with sand or other similar materials of the space from which the coal is removed. This system originated in the United States and is now successfully practised in portions of Germany, Austria, Belgium, and France.

I.—Mine Supervision and Inspection.—1.—We cannot too strongly emphasise the fact that thorough discipline about the mine is absolutely essential to safety, and that thorough discipline can be brought about only through the hearty co-operation of the operators, the miners, and the State.

2.—We are of the opinion that the responsibility for safety in the mine should primarily rest with some person, such as the manager or superintendent, clothed with full authority; and that such person can greatly facilitate the attainment of safety through the employment of a sufficient number of foremen, and also of one or more inspectors whose special duty it shall be to see that the regulations are strictly enforced.

3.—The State cannot exercise too much care concerning the experience, technical training, and selection of its inspectors. Their positions should be made independent of all considerations other than that of efficiency; and their continuance in the service should be co-existent with good behaviour and proper discharge of official duty.

J.—Training for Mine Foremen, Inspectors, etc.—We are of the opinion that the cause of both safety and efficiency in coal mining in the United States would be greatly aided through the establishment and maintenance in the different coal regions of special schools for the training of fire bosses, mine foremen, superintendents, and inspectors. The instruction in such schools should be practical rather than theoretical.

The work of these schools would supplement most effectively that of the colleges already established in many parts of the country for the more thorough training of mining engineers."—*Science and Art of Mining*, Dec. 26, 1908, pp. 226-7. (A. R.)

SURVEYS AND RECORDS.—"In few districts are such careful surveys made of both surface and underground workings as here. Ample corps of college-trained engineers and geologists are employed to observe and record every important phase of development and geology.

The system of the Cleveland Cliffs Co. is as follows: Underground surveys of the drifts are made with a transit and 150 ft. steel tape, using for stations, punched horseshoe nails driven in an overhead timber or plug. Only horizontal positions are determined by transit, so any elevations must be measured separately by levelling from the nearest bench-mark. Mine maps are made on a scale of 50 ft. per in. (on paper rules with 200 ft. squares), and surface exploration maps on a scale of 200 ft. per in. A separate tracing is made of each mine level, and on this sheet the geological formation is filled in by the geologist.

An annual record book is made for each mine, containing photo prints of any new surface improvements and of the underground plans and sections. The maps are photographed from tracings, held against a glass window, by a camera with 15 in. × 17 in. plates. The negatives, thus made, show dark lines on a white background when printed, and the camera focus is set to make the negative maps of a uniform scale of 200 ft. per in. A reducing plate holder is used, so that a plate need be no longer than is necessary to cover the given map. The prints are all made on 15 in. × 17 in. blue-print sheets, and the smaller plates are printed within black bordered frames, so as to show white margins.

For the Mesabi open pits, the Oliver Co. pursues the following survey system: Before the contractors begin stripping, the whole ore area is blocked out in 100 ft. squares and stakes set across the deposit on the 100 ft. lines at 20 ft. intervals. The level of each station is then taken, and by holding a tape between neighbouring stakes to make the longitudinal 20 ft. marks, the remaining levels can be observed at the corners of each 20 ft. square. When the drift has been removed down to the ore, the new level of each 20 ft. point is observed, and then the total volume of stripping can be accurately calculated. Recently the company has started doing some stripping on company account, and here levels are taken only at 40 ft. intervals.

Subsequently the ore removed during any given period can be easily calculated by securing new elevations of the same points used in the stripping survey. The ore reserves were formerly computed from the drill hole records by treating each body as a whole. Owing to the fact that the Mesabi ore runs in thick layers of large area, but differing analyses, it has been found possible, with the same drill records, to estimate the volume of each ore layer separately, and this has made the results of the reserve calculations much more instructive and useful."—*Mining Science*, Dec. 3, 1908, p. 447. (A. R.)

MISCELLANEOUS.

THE ETHER OF SPACE.—"The following is an abstract of the lecture delivered by the author before the Royal Institution of Great Britain.

Thirty years ago Clerk Maxwell gave in this place a remarkable address on 'Action at a distance.' Most natural philosophers hold, and have held, that action at a distance across empty space is impossible —in other words, that matter cannot act where it is not, but only where it is. There can be no attraction across really empty space. And even when a material link exists so that the connection is obvious the explanation is not complete, for when the mechanism of attraction is understood it will be found that a body really only moves because it is pushed by something from behind. Why the whole of a rod should follow, when one end is pulled, is a matter requiring explanation, and the only explanation that can be given involves, in some form or other, a continuous medium connecting the discrete and separated particles or atoms of matter.

When a steel spring is bent or distorted, what is it that is really strained? Distortion of a spring is really distortion of the ether. All stress exists in the ether. Matter can only be moved. Contact does not exist between the atoms of matter as we know them. Matter acts on matter only through the ether. But whether matter is a thing utterly distinct and separate from the ether, or whether it is a specifically modified portion of it, are questions demanding, and I may say, in process of receiving, answers.

It has been said, somewhat sarcastically, that the ether was made in England. First of all, Newton recognised the need of a medium for explaining gravitation. The force with which the moon is held in its orbit would be great enough to tear asunder a steel rod 400 miles thick, with a tenacity of 30 tons per square inch. Such a force necessarily implies enormous tension or pressure in the medium. Maxwell calculates that the gravitational stress near the earth, which we must suppose to exist in the invisible medium, is 3,000 times greater than what the strongest steel could stand; and near the sun it should be 2,500 times as great as that.

I pass to another function, the electric and magnetic phenomena displayed by the ether. Faraday conjectured that the same medium which is concerned in the propagation of light might also be the agent in electro-magnetic phenomena. One more function is now being discovered; the ether is being found to constitute matter—an immensely interesting topic, on which there are many active workers at the present time. I will make a brief quotation from your present professor of natural philosophy (J. J. Thomson). 'The *whole* mass of any body is just the mass of ether surrounding the body which is carried along by the Faraday tubes associated with the atoms of the body. In fact, all mass is mass of the ether; all momentum, momentum of the ether; and all kinetic energy, kinetic energy of the ether. This view, it should be said, requires the density of the ether to be immensely greater than that of any known substance.' Not till last year, however, did I realise what the density of the ether must really be, compared with that modification of it which appeals to our senses as matter, and which for that reason engrosses our attention. If I have time I will return to that before I have finished.

I shall now attempt to illustrate some relations between ether and matter. The question is often asked, is ether material? This is largely a question of words and convenience. Undoubtedly, the ether belongs to the material or physical universe, but it it is not ordinary matter. I should prefer to say it is not 'matter' at all. It may be the substance or substratum or material of which matter is composed, but it would be confusing and inconvenient not to be able to discriminate between matter, on the one hand, and ether on the other. If you tie a knot on a bit of string, the knot is composed of string, but the string is not composed of knots. If you have a smoke or vortex ring in the air, the vortex ring is made of air, but the atmosphere is not a vortex ring; and it would be only confusing to say that it was. The essential distinction between matter and ether is that matter *moves*, in the sense that it has the property of locomotion and can effect impact and bombardment; while ether is *strained*, and has the property of exerting stress and recoil. But now comes the question, how is it possible for matter to be composed of ether? How is it possible for a solid to be made out of fluid? A solid possesses the properties of rigidity, impenetrability, elasticity, and such like; how can these be imitated by a perfect fluid such as the ether must be? The answer is, they can be imitated by a fluid in motion—a statement which we make with confidence as the result of a great part of Lord Kelvin's work. It may be illustrated by a few experiments. A wheel of spokes, transparent or permeable when statinary, becomes opaque when revolving, so that a ball thrown against it does not go through, but rebounds. The motion only affects permeability to matter; transparency to light is unaffected. A silk cord hanging from a pulley becomes rigid and viscous when put into rapid motion; and pulses or waves which may be generated on the cord travel along it with a speed equal to its own velocity, whatever that velocity may be, so that they appear to stand still. This is a case of kinetic rigidity; and the fact that the wave-transmission velocity is equal to the rotary speed of the material is typical and important, for in all cases of kinetic elasticity these two velocities are of the same order of magnitude. A flexible chain, set spinning, can stand up on end while the motion continues. A jet of water at sufficient speed can be struck with a hammer, and resists being cut with a sword. A spinning disc of paper becomes elastic like flexible metal, and can act like a circular saw. If the ether can be set spinning, therefore we may have some hope of making it imitate the properties of matter, or even of constructing matter by its aid. But how are we to spin the ether? Matter alone seems to have no grip of it. I have spun steel discs, a yard in diameter, 4,000 times a minute, have sent light round and round between them, and tested carefully for the slightest effect on the ether. Not the slightest effect was perceptible. We cannot spin ether mechanically, but we can vibrate it electrically, and every source of radiation does that. An electrified body, in sufficiently rapid vibration, is the only source of ether waves that we know, and if an electric charge is suddenly stopped it generates the pulses known as X-rays as the result of the collision. Not speed, but sudden change of speed, is the necessary condition for generating waves in the ether by electricity. We can also infer some kind of rotary motion in the ether, though we have no such obvious means of detecting the spin as is furnished by vision for detecting some kinds of vibration. It is supposed to exist whenever we put a charge into the neighbourhood of a magnetic pole. Round the line joining the two, the ether is spinning like a top. I do not say it is spinning fast—that is a question of its density; it is, in fact, spinning with excessive slowness, but it is spinning with a definite moment of momentum. J. J. Thomson's theory makes its movement of momentum exactly equal to em, the product of charge and pole; the charge being measured electrostatically and the pole magnetically. How can this be shown experimentally? Suppose we had a spinning top enclosed in a case, so that the spin was unrecognisable by ordinary means—it could be detected by its gyrostatic behaviour to force. If allowed to 'precess' it will respond by moving perpendicularly to a deflecting force. So it is with the charge and the magnetic pole. Try to move the charge suddenly and it immediately sets off at right angles. A moving charge is a current, and the pole and the current try to revolve round one another—a true gyrostatic action due to the otherwise unrecognisable etherial spin.

Lastly, we have the fundamental property of matter called *inertia*, which, if I had time, I would show could be explained electromagnetically, provided the etherial density is granted as of the order 10^{12} gm. per cubic centimetre. The elasticity of the ether would then have to be of the order 10^{33} C.G.S., and if this is due to intrinsic turbulence, the speed of the whirling or rotational elasticity must be of the same order as the velocity of light. And so, on our present view, the intrinsic energy of constitution of the ether is incredibly and portentously great; every cubic millimetre of space possessing what, if it were matter, would be a mass of a thousand tons, and an energy equivalent to the output of a million horse-power station for 40 million years.

The etherial tension near the earth's surface, required to explain gravity by its rate of variation, is of the order 6×10^{11} C.G.S. units. The tension near the sun is 2,500 times as great. With different spheres in general it is proportional to the density and to the superficial area. Hence, near a bullet 1 in. in diameter it is of the order 10^{-6}, and near an atom or electron about 10^{-21}. In order to set up a tension equal to the critical, or presumably disruptive, stress in the ether (10^{33} C.G.S.), a globe of the density of the earth would have to have a radius of eight light years. In order to generate a velocity of free fall under gravity equal to the velocity of light, a globe of the earth's density would have to be equal in radius to the distance of the earth from the sun, or, say, 26,000 times the earth's radius. The whole visible universe with a parallax of $\frac{1}{1000}$ second of arc, estimated by Lord Kelvin as the equivalent of 10^9 suns, would be quite incompetent to raise etherial tension to the critical point 10^{33} C.G.S. unless it were concentrated to an absurd degree; but it could generate the velocity of light with a density comparable to that of water. If the average density of the above visible universe (which may be taken as $1{\cdot}6\times10^{-23}$ gm. per c.c.) continued without limit, a disruptive tension of the ether would be reached when the radius was comparable to 10^{13} light years; and the velocity of light would be generated by it when the radius was 10^7 light years. But heterogeneity would enable these values to be reached *more* easily. It is noteworthy how exceedingly small is the average or aggregate density of matter in the visible region of space; and Lord Kelvin has shown that throughout space in general it must be smaller still—in fact, ultimately infinitesimal. The estimated density of 10^{-23} C.G.S. means that the visible cosmos is as much rarer than a vacuum of a hundred millionths of an atmosphere, as that vacuum is itself rarer than lead. It is, of course, because ordinary masses of matter likewise consist of separated particles, with great intervening distances in proportion to their size, that we are able to assert the minute aggregate density of ordinary stuff—such as water or lead—as compared with the continuous medium of which all particles are supposed to be really composed. The fundamental medium itself must be of uniform density everywhere, whether materialised or free.

Explanatory Remarks concerning the Density of Ether.—The ether inside matter is just as dense as the ether outside, and no denser. A material unit—say, an electron—is only a peculiarity or singularity of some kind in the ether itself, which is of perfectly uniform density elsewhere. What we sense as matter is an aggregate or grouping of an enormous number of such units. How, then, can we say that the matter is millions of times rarer or less substantial than the ether of which it is essentially composed? Those who feel any difficulty here should bethink themselves of what they mean by the average or aggregate density of any discontinuous system, such as a powder, or a gas, or a snowstorm, or a cloud, or a milky way. If it be urged that it is unfair to compare an obviously discrete assemblage like the stars with an apparently continuous substance like air or lead, the answer is that it is entirely and accurately fair; since air, and every other known form of matter, is essentially an aggregate of particles, and since it is always their average density that we mean. We do not even know for certain their individual atomic density. The phrase 'specific gravity or density of a powder' is ambiguous. It may mean the specific gravity of the dry powder as it lies, like snow, or it may mean the specific gravity of the particles of which it is composed, like ice. So also with regard to the density of matter, we might mean the density of the fundamental material of which its units are made—which would be ether—or we might, and in practice do, mean the density of the aggregate lump which we can see and handle—that is to say, of water, or iron, or lead, as the case may be. In saying that the density of matter is small, I mean, of course, in the last, the usual sense. In saying that the density of ether is great, I mean that the actual stuff of which these highly porous aggregates are composed is of immense, of well-nigh incredible density. It is only another way of saying that the ultimate units of matter are few and far between—*i.e.*, that they are excessively small as compared with the distances between them; just as the planets of the solar system, or worlds in the sky, are few and far between—the intervening distances being enormous as compared with the portions of space actually occupied by lumps of matter. Here it may be noted that it is possible to argue that the density of a *continuum* is necessarily greater than the density of any disconnected aggregate, certainly of any assemblage whose particles are actually composed of the material of the *continuum*. Because the former is 'all there,' everywhere, without break or intermittence of any kind; while the latter has gaps in it—it is here, and there, but not everywhere.

We do not depend on that sort of argument, however; what we depend on is experimental measure of the mass, and mathematical estimate of the volume, of the electron For calculation shows that however the mass be accounted for, whether electrostatically, or magnetically, or hydrodynamically, the estimate of ratio of mass to effective volume can differ only in a numerical coefficient, and cannot differ as regards order of magnitude. The only way out of this conclusion would be the discovery that the negative electron is not the real or the main matter unit, but it is only a subsidiary ingredient, whereas the main mass is the more bulky positive charge. That last hypothesis, however, is at present too vague to be useful. Moreover, the mass of such a charge would in that case be unexplained, and would need a further step, which would probably land us in much the same sort of etherial density as is involved in the estimate which I have based on the more familiar and tractable negative electron. It may be said, why assume any finite density for the ether at all? Why not assume that, as it is infinitely continuous so it is infinitely dense—whatever that may mean—and that all its properties are infinite? This might be possible were it not for the velocity of light. By transmitting waves at a finite and measurable speed, the ether has given itself away, and has let in all the possibilities of calculation and numerical statement. Its properties are thereby exhibited as essentially finite, however infinite the whole extent of it may turn out to be."—SIR OLIVER LODGE.—*Electrical Engineer*, Aug. 28, 1908, p. 307. (H. A. W.)

Reviews and New Books.

THE MINE EXAMINER AND PROSPECTOR'S COMPANION, an illustrated field book. By G. W. MILLER, E.M., C.E. Fifth edition. Revised, 12s. 6d. (London: Edward Walker, London Wall, E.C.)

"When a book has reached its fifth edition, it needs no further recommendation; on the other hand, there are new men continually coming into the field to whom the very existence of so well-proved a pocket book is unknown, and in their interest the new edition of Miller's Field Book claims attention. In a prefatory note, the publishers state that "more than 3,000 copies are now in the hands of the leading consulting mining engineers of the United States, Europe, and other countries, and that it has been adopted as a standard text-book on mine examination." The volume is intended for the use of mining students, engineers, prospectors, mining men, and the general reader who takes an interest in such matters as mining geology, mineralogy, ore milling practice, testing and assaying, reduction processes, etc. Prospectors who carry the book in their kits will find great use for the long analytical tables of minerals, which are so elaborately descriptive that they must greatly assist in identifying the various minerals which come under notice. The pocket-book is well illustrated with diagrams, and is in a soft leather binding."—*The Mining Journal*, London, Nov. 4, 1908, p. 510. (W. A. C.)

WEIGHTS AND MEASURES. By JAMES ROBERTS, Barrister-at-Law. Third edition. 451 pp., 8½ × 5½. (Charles Knight & Co., Ltd., London; and Hanna & Neale, Dublin.) Price, 8s. net.

"In this volume the author has succeeded in the two-fold task of explaining the general principles of construction and management of weight and measuring machines, and of setting forth the Weights and Measure Acts, the Board of Trade Regulations, and the statutory provisions relating to the sale or checking of various commodities by weighing or measure. It has already approached very nearly to being the standard work on the subject, and this third edition will go far towards establishing it as the authoritative hand-book of commercial weights and measures. The scope extends through all commodities from bread to coal and from coal to drugs; solids and liquids with their multiplicity of units are all assigned to their appropriate conditions, side by side with the law concerning them; and British and metric tables and systems are dealt with in an uncontroversial and judicial spirit that, in a work of this kind, is extremely commendable. In stating that the 'metric' system is sometimes called the 'decimal' system, the author is, no doubt, correct; but it might perhaps have been well to explain that this custom is incorrect. The 'metric' system is identified with the mètre as a fundamental unit; but the 'decimal' system is applicable to any unit whatever, and it enables us, for example, in this country to deal in decimals with any of our rich supply of units that happens to be convenient. Thus the metric system is *a* decimal system, but it is not *the* decimal system. This is a distinction of extreme importance. To all traders and inspectors the coordinate legal and general information relating to specific commodities must be of great value."—*The Times Engineering Supplement*, Dec. 2, 1908, p. 19. (W. A. C.)

MODERN PRACTICE IN MINING. Vol. I., Coal: Its Occurrence, Value and Methods of Boring. By R. A. S. REDMAYNE, M.Sc., etc. (London: Longmans, Green & Co., 39, Paternoster Row, E.C.)

"In his preface to his volume Prof. Redmayne makes the important announcement that it is the first of a series, constituting a complete work on modern practice in mining. We gather that the five volumes are already in hand, of which the first is now issued; the second will deal with the sinking and equipment of colliery shafts; the third with the methods of working coal; the fourth with the ventilation of coal mines, and the fifth with mechanical engineering as applied to collieries. Other volumes, on metalliferous mining, are contemplated, as well as one on mine surveying, the latter the work of Mr. H. Briggs, B.Sc., A.R.S.M., and will be added to this series from time to time, so that the entire work will embrace a complete scheme of instruction in coal, ore, and stone mining, and yet allow of any one volume being obtained without committing its owner to the acquisition of any others of the series. The unquestionable value of Prof. Redmayne's extended experience lies in the fact that it enables him to blend theory with practice, a feature of his work which is abundantly manifest in the volume under review. In the first three chapters he enters upon the geological aspects of coal mining, disclaiming, however, any attempt to provide a substitute for the serious study of constructive geology, which constitutes so important a part of the equipment of the mining engineer. He gives in a concise yet comprehensive form such geological and chemical facts as are of special interest from the coal miner's point of view. The remaining three chapters of the work are devoted to prospecting and boring for coal. Considerations of space prohibit an extended review of this portion of his work, but it may be noted that the theoretical and practical aspects of the object are presented with admirable lucidity, aided by numerous illustrations and diagrams of a useful type."—*Iron and Coal Trades Review*, Dec. 4, 1908, p. 2,418. (A. R.)

STRENGTH OF MATERIALS. By ARTHUR MORLEY. 7s. 6d. (London: Longmans, Green & Co.)

"As an educational work Professor Morley's treatise has distinctive claims that must strongly appeal to students. Too many authors, in what might be styled the field of academical literature, confine themselves to theory. Examples, which are the application of theories, are omitted; or, if inserted, the answer are left to be found and verified by the reader: or final results, if supplied, yet leave to the student the arduous labour of discovering the intricate calculations by which they are reached. In 'Strength of Materials' the author does the right thing, and it cannot be too strongly impressed on authors that it is the right thing—namely, to insert, fully worked out, examples in illustration of the theoretical principles laid down. We are convinced that by this method the value of the work before us has been greatly enhanced, and its sale will be proportionately increased. The subject matter dealt with by the author comprises—elastic stress and strain, mechanical properties of metals, resilience and fluctuating stress, theory of bending, stresses and deflection in beams, effects of bending, twisting, pipes, cylinders, discs, curved bars and flat plates, vibrations and critical speeds, testing machines, apparatus and methods, etc. In discussing the strength of rotary discs and cylinders, the bending of curved bars, the the strength of unstayed flat plates, and the critital speeds of running machinery, the author introduces matter that has so far received little attention in other text-books. The 487 pages include 248 diagrams, tables, and index, and is, as might be expected, excellently printed and published."—*The Mining Journal*, London, Dec. 19, 1908, p. 792. (W. A. C.)

MANUAL OF UNDERGROUND SURVEYING. By LOYAL W. TRUMBULL. 251 pp., illustrated; $6\frac{1}{4} \times 9\frac{1}{4}$ in. Cloth; price, 3 doll. (New York and London: Hill Publishing Co., 1908.)

"Contents: Instruments. Meridian. Underground practice. Carrying the meridian underground. Survey of secondary openings. Record of the survey. Uses of the mine maps. Making the map. Map filling. Borehole surveys. Methods of various engineers. United States deputy mineral surveyor's examination.

This is a good book. The author evidently lacks experience in literary work, which deprives his book of the scholarly effect that such a treatise ought to have. His long quotations from other writers, however pertinent and satisfactory in their instruction as to the subject, are apt to be somewhat irritating to the reader. But these are trifling criticisms. The essential thing is that we have here a complete, intelligent and practical treatise on mine surveying, especially as practised in America, that is in all respects up to date. It comprises many novel features that will be warmly appreciated, some of which are mentioned in the table of contents, summarised above. The author is an engineer who has done personally the things that he writes about."—*Engineering and Mining Journal*, Dec. 5, 1908, p. 1,115. (W. A. C.)

Adie, R. H. Junior Chemistry. Cr. 8vo. pp. viii.—226. *W. B. Clive.* 2s. 6d.

Chemical Manufacturers' Directory for 1909. Cr. 8vo., pp. 189. *Office.* Net 6s.

Hatch, F. H. Text Book of Petrology. 5th edition. Revised and re written. Cr. 8vo., pp. 420. *Sonnenschein.* Net 7s. 6d.

Heidenreich, E. L. Engineer's Pocket Book of Reinforced Concrete. 12mo, lr. *Spon.* Net 12s. 6d.

Jones, Lionel M. Practical Physics. Cr. 8vo., pp. 338. *Longmans.* 3s.

Law, Edward F. Alloys and their Industrial Applications. 8vo., pp. 286. *C. Griffin.* Net 12s. 6d.

Lewis, E. J. The Elements of Organic Chemistry. Cr. 8vo., pp. viii.—224. *W. B. Clive.* 2s. 6d.

Maclaren, J. M. Gold, its Geological Occurrence and Geographical Distribution. Illustrated. *London Mining Journal.* Net 25s.

Mathewson, C. H. First Principles of Chemical Theory. 8vo. *Chapman & Hall.* Net 4s. 6d.

McCall, A. G. The Physical Properties of soils. Cr. 8vo. *K. Paul.* Net 2s. 6d.

Poynting, J. H., and Thomson, J. J. A Text Book of Physics. Sound. 5th edition. Carefully revised. 8vo., pp. 176. *C. Griffin.* 8s 6d.

Practical Coal Mining. By leading Experts in Mining and Engineering. Under the Editorship of W. S. Boulton. Div. vol., 6. 8vo. *Gresham Publishing Co.*

Richards, E. H. Laboratory Notes on Industrial Water Analysis. 8vo. *Chapman & Hall.* Net 2s. 6d.

Science in Modern Life. A Survey of Scientific Development, Discovery and Invention, and their Relations to Human Progress and Industry, Prepared under the Editorship of J. R. Ainsworth Davis. Vol. 2. 4to, pp. viii.—187. *Gresham Publishing Co.*

Senter, George. Outlines of Physical Chemistry. Cr. 8vo., pp. 388. *Methuen.* 3s. 6d.

Wade, John. Introduction to the Study of Organic Chemistry. New and enlarged edition Cr. 8vo. *Sonnenschein.* Parts 1 and 2, net 5s. Part 3. net 4s. 6d.

Spitta, Edmund J. Microscopy. The Construction, Theory and Use of the Microscope. Illustrated. 8vo., pp. 524. *J. Murray.* Net 12s. 6d.

Trentham, W. H., and Saunders, James. Modern Methods of Sewage Disposal. A Practical Handbook for the Use of Members of Local Authorities and their Officials. Cr. 8vo., pp. viii.—60. *Sanitary Publishing Co.* Net 2s. 6d.

Wiley, Harvey W. Principles and Practice of Agricultural Analysis. 2nd edition. Revised and enlarged. Vol. 2. Fertilisers and Insecticides. 8vo. *Williams and Norgate.* Net 20s.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 46/09. Matthew Jordan Adam. Improvements in covers for protecting amalgamating plates, extractor boxes, launders and the like. 28.1.09.

(P.) 47/09. John Coad. Improvements in rock drilling machine chucks. 30.1.09.

(P.) 48/09. James Edward Thomas. Improvements in or relating to lathes for cutting sheet zinc and the like. 30.1.09.

(P.) 49/09. James Edward Thomas. Improvements in the linings of tube and like mills. 30.1.09.

(P.) 50/09. Gilbert Kemp. A device for an automatic coupling for rolling stock. 30.1.09.

(P.) 51/09. Henry James Shedlock Heather. Protective device for oil cooled electrical transformers. 30.1.09.

(P.) 52/09. Ludwig Hampel. Improvements in the formation or construction and maintenance of slimes dams. 1.2.09.

(P.) 53/09. James Snodgrass. Improvements in apparatus for the manufacture of cyanide. 1.2.09.

(P.) 54/09. John Johnstone. Improvements in jockeys or rope grips for mechanical haulage arrangements. 1.2.09.

(P.) 55/09. Rollo Bowman Ballantine. An improved process for treatment of ores containing high value metals. 2.2.09.

(P.) 57/09. William Arthur Caldecott. A new or improved substance for precipitating metals from their solutions. 3.2.09.

(P.) 58/09. Robert Hutchison Anderson. Improvements in rock drills. 3.2.09.

(P.) 59/09. James Clayton (1), George Ritchie Lambie (2). Improvements in furnace grates. 3.3.09.

(P.) 60/09. George Reid McLean. Improvements in or relating to endless rope mechanical haulage and automatic or self-discharging trucks therefor. 4.2.09.

(P.) 61/09. Samuel Robinson. Improvements in means for packing reciprocating pistons. 3.2.09.

(P.) 62/09. Anthony Maurice Robeson. Improved method for precipitating metals from solutions. 3.2.09.

(P.) 63/09. James Gwynn. Improvements in jockeys for mechanical haulage. 8.2.09.

(P.) 65/09. Alfred James. Improvements in automatic filters. 12.2.09.

(P.) 66/09. Colin Armstrong Ferguson (1), Norman Mathieson (2). Improvements in pulp samplers. 12.2.09.

(P.) 67/09. Harry August Stockman (1), Sidney Frederick Allen (2). Improvements in axle bearings for mine trucks and the like. 12.2.09.

(P.) 68/09. Robert Telford Hetherington. Improvements in covers for battery plates, extractor boxes and the like. 15.2.09.

(P.) 69/09. Francois Guy Comte de Ferrières (1), Benjamin Waite (2). Improvements in hydraulic classifying and concentrating apparatus. 16.2.09.

(C.) 70/09. Anthony Maurice Robeson (1), Andrew Davidson (2). Improvements in centrifugal pumps. 16.2.09.

(P.) 71/09. Philip Campbell Hill Glover (1), Alfred Thomas Banks (2). Improvements in rock drilling machine bits and hand rock drills. 17.2.09.

(P.) 72/09. Peter Milne Duncan. Improvements in means or devices for taking samples from streams of liquids or liquids mixed with solids or comminuted matter in the dry state. 17.2.09.

(P.) 73/09. Frederick Harsman. Improvements in and connected with batteries, stamp mills and the like, more especially those in connection with the extraction of gold and base metals. 18.2.09.

(P.) 74/09. Charles Reeves (1), Thomas Edwards (2). Automatic tipping apparatus for discharging residue from all kinds of trucks or carriages. 18.2.09.

(P.) 75/09. Charles Harrison Dixon. Improvements in tappets for gravity stamps. 18.2.09.

(C.) 77/09. Michael Neilson. Improvements in means for use in dumping of the contents of truck or like vehicles on mine dumps or like depositing sites. 18.2.09.

(P.) 78/09. Charles Cheers Wakefield. Improvements in or relating to the treatment of calcium carbide. 19.2.09.

(C.) 79/09. Anton Steinlein (1), Badische Maschinenfabrik und Eisengiesserei, vormals G. Sebold & Sebold & Neff (2). Improvements in apparatus for separating or washing solid substances. 19.2.09.

(P.) 81/09. Ernest Holroyd Martin. Improvements in pulp diverting means for amalgamating plates. 20.2.09.

(P.) 82/09. Vivian Thornton Moys. Apparatus for utilising the heat generated by stoves, furnaces and the like, otherwise wasted in chimney. 23.2.09.

(C.) 84/09. Charles Edward Martin. Improvements in or relating to rotary pumps or the like. 23.2.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices

Bowyer, Jas., *to* P. O. Box 131, East Rand.

Christopherson, E. H., *l/o* Johannesburg; West Rand Consolidated Mines, Ltd., P. O. Box 38, Krugersdorp.

Coaton, A. A., *l/o* Knights; Crown Deep, Ltd., P. O. Box 102, Fordsburg.

Dowsett, C. W., *l/o* Knights; Knights' Central G. M. Co., Ltd., P. O. Box 91, Germiston.

Hahn, W. A. A., *l/o* Denver; South-West Africa Co., Grootfontein, via Swakopmund, German South-West Africa.

Milligan, J. T., *l/o* Johannesburg; New West Bonanza G. M. Co., Ltd., P. O. Box 137, Klerksdorp.

Mossop, W. C., *l/o* Johannesburg: Treasury Gold Mines, Ltd., Cleveland.

Murdoch, R., *l/o* Gwanda: Kimberley Reefs, Mazoe, Rhodesia.

Rand, E. T., *l/o* East Rand; P. O. Boksburg North.

Smith, S. A., *l/o* Johannesburg; Modderfontein "B" G. M. Co., Ltd., P. O. Box 14, Brakpan.

Weston, E. M., *l/o* Johannesburg; McCreedy Tins, Ltd., Mbabane, Swazieland.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

Vol. IX. MARCH, 1909. No. 9.

Proceedings
AT
Ordinary General Meeting,
March 20, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, March 20th, Mr. R G. Bevington (President), in the chair. There were also present:—

50 Members: Dr. J. Moir, Messrs. A. McA. Johnston, W. R. Dowling, A. Richardson, C. B. Saner, G. O. Smart, Prof. G. H. Stanley, J. E. Thomas, A. Whitby, H. A. White, Prof. J. A. Wilkinson, W. A. Caldecott, W. Cullen, F. F. Alexander, E. Blume, H. O. Bowen, W. Brown, P. Carter, F. T. Chapman, A. A. Coaton, M. H. Coombe, W. M. Coulter, L. Evans, A. D. Gilmour, Dr. J. McC. Henderson, A. Heymann, C. G. James, A. J. Johnson, J. Kennedy, R. N. Kotzé, G. A. Lawson, Hy. Lea, J. Lea, G. Melvill, S. Newton, T. T. Nichol, J. F. Pyles, L. J. Robinson, N. Rogers, O. D. Ross, H. A. Scarf, W. Sharp, F. L. Simmons, R. Stokes, H. Taylor, J. A. Taylor, O. Tonnesen, A. D. Viney, J. P. Ward, and F. W. Watson.

10 Associates: Messrs. J. Bowyer, J. Chilton, J. Cronin, M. Dennehy, H. Stadler, G. G. Thomas, W. E. Thorpe, W. Waters, J. Whitehouse and J. A. Woodburn.

17 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the February *Journal*, were confirmed.

NEW MEMBERS.

Messrs. H. M. Coombe and P. L. Edwards were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been duly elected, as follows:—

Adams, Hirzel Allen, Village Main Reef G. M. Co., Ltd., P. O. Box 1091, Johannesburg. Assistant Surveyor.
Hövig, Peter, Benkoelen, Sumatra, Dutch East Indies. Government Mining Engineer.
Mehliss, Max, M.D., P. O. Box 1076, Johannesburg. Medical Superintendent, Rietfontein Lazaretto.
Tonnesen, Karl Oscar, C.E., P. O. Box 6647, Johannesburg. Construction Engineer, Knights Central, Ltd.
Wernher, Sir Julius C., Bart., 1, London Wall Buildings, London, E.C.

The Secretary: Since the last meeting of the Society the following associates have been admitted:—

Bird, Frank Arthur, Ancona Cottage, Ivy Mine, Barberton. Cyanide Manager.
Kay, Herbert George, P. O. Box 182, Premier Mine, Cullinan. Surveyor.
Nakamoto. Hidehiko, Sado Gold Mine, Aikawa, Sado, Japan. Sub-Manager.

The President: It is very satisfactory to see that our Society has now got so far as Japan. Its ramifications are evidently extending all over the world.

General Business.

The President: There is one announcement under the head of general business, and that is with regard to our annual dinner. The actual date was not settled in time to place it on the agenda paper for this meeting, but it has since been settled as Saturday, the 24th of April, that being practically the only convenient date. The previous Saturday is our general meeting, and we do not like to depart from our usual practice and give up our meeting for anything. The Saturday preceding that is the Saturday before Easter, which is hardly a suitable date. Anyone requiring tickets can obtain them from our Secretary to-night. The price is one guinea, and I sincerely hope that as many members as possible will attend and make our dinner the success it always has been.

GOLD LAW REGULATIONS.

Mr. F. F. Alexander (*Member*): I would like to ask, under general business, whether the Council has done anything to advise the Government with regard to the covering of our extractor boxes, and whether the Council has taken any steps to advise the industry as to whether the whole of the boxes shall be covered or not?

The President: I may say that the Council considered the question in the first instance, and they decided that it was rather a matter which should emanate from the Government, and from the Consulting Engineers of the various groups. The Council felt that it could not take any action as it was a matter of law and the carrying out of regulations. We considered that it was entirely a matter between the Government and the various Consulting Engineers.

Mr. F. F. Alexander: I have brought this subject up because some of the Government officials express ignorance on it. I think this Society should take active steps in this case, and I believe it would facilitate things. Personally, I do not think that the whole of the boxes need be covered up.

Mr. R. N. Kotze (*Government Mining Engineer*): I think it would fill a useful purpose since so many of our members are interested in this question, if a note were published in the Society's *Journal* to the effect that the Government will not insist upon weak extractor boxes being covered. The question as to which are weak and which are strong extractor boxes will be left to the Inspectors of Mines.

Mr. F. F. Alexander: I had occasion to go into the question to-day. It is, in my opinion, an irksome law which enforces the covering of the whole of the extractor boxes, or even the whole of the strong boxes. I believe it is not too late now for this Society to take action and approach the Government, so as to guide them as to what is really right in the matter.

Mr. H. A. White (*Member of Council*): I might say that, at my instance, the Council of the Society appointed a sub-committee to go into this question and review these regulations, a copy of which was submitted confidentially to them by the Government Mining Engineer for the purpose. We found certain small details which might have been improved, but we were informed that these regulations were the result of a compromise between the Government and a committee appointed by the Mine Managers' Association. We were satisfied that the gentlemen responsible for the technical control of our works had agreed to accept these regulations, and under these circumstances we decided that the alterations we might have put forward were perhaps better left alone. We, therefore, wrote to the Government Mining Engineer to the effect that we had considered the matter, and that having been informed that these regulations had been agreed to by the Mine Managers' Association and others who are responsible, we decided to make no suggestion.

The President: Yes, I may say that is exactly what took place.

Mr. F. F. Alexander: I still assert that we have time to represent our views. Why put these points as being trivial? I think they are not trivial, and that they affect us, as workers in the industry, very much. I think the Council should formulate some scheme, and represent their scheme through the proper channel to the Government.

Mr. H. A. White: It was suggested that we should ask members to bring forward descriptions of appliances, but it was agreed by the Council that these matters were better left in the hands of the Consulting Engineers of the groups.

Mr. F. F. Alexander: Then I understand the Council have really shelved a matter of great importance?

The President: I do not think you can say that, Mr. Alexander. The matter is one of law.

Mr. F. F. Alexander: I do not think it is a matter of law until the Government enforces it, and I cannot believe the Government wishes to hamper us with any law that is irksome.

The President: It became law on the 2nd of January. Of course, Mr. Alexander, it is quite competent for you, as a member, to move a resolution instructing your Council to take action.

Mr. F. F. Alexander: I move a resolution that the Council be instructed to take immediate steps to investigate the matter, and to combine with the Mine Managers' Association and through them the Chamber of Mines, in drafting arrangements whereby these extractor boxes may be wholly or partly covered, subject to the approval of the Government Inspector.

Mr. J. E. Thomas (*Member of Council*): I beg to point out that what will suit one mine will not suit another. I think it should easily be arranged with the Inspectors of Mines in each particular district in which a mine is situated.

The President: Will anyone second Mr. Alexander's resolution?

Mr. W. Cullen (*Past-President*): I will second it.

The President: Are there any amendments or any discussion?

Mr. A. McArthur Johnston (*Vice-President*): I beg to move that the action of the Council be confirmed.

Mr. G. O. Smart (*Member of Council*) seconded the amendment, which was carried by 24 votes to 7.

COLOURED WRAPPERS FOR EXPLOSIVES.

Mr. W. Cullen (*Past-President*): The suggestion contained in Mr. Brett's letter has been made before, but that does not detract from its value; indeed, to a certain extent, it has already been adopted in other directions, *vide* the white wires attached to electric detonators for use in coal mines. Anything, however, which will decrease the risk of accident is worth investigating, consequently I made up a lot of dummy cartridges wrapped with paper of different colours, and tried them underground by ordinary candlelight. Speaking generally, under ordinary working conditions, it was found that red and green showed up best against the rock, so I have accordingly ordered a quantity of paper of these colours in order to try the experiment on a more extensive scale. The present wrappers are found to be distinctive enough for coal mining, but it stands to reason the colours would require modification for other conditions. For instance, white cartridges would show up best against haematite. It will take some considerable time before the paper I have just referred to will arrive, for a pigment will require to be found which will not affect the chemical stability of the explosive and, therefore the heat test, which is the index of chemical stability. This is far from being an easy matter, and I could not suggest any pigment straight off.

The paper for cartridge wrappers must be specially pure for the reason I have just indicated, and I should not be at all surprised if that is the main reason why coloured wrappers have never been used before. Of course, where a stick of explosive has been stripped of its wrapper by the force of the explosion there is nothing to be gained by having a distinctive colour, and I should therefore like to know from practical men whether it is more common to come across the stripped or the unstripped cartridge.

The President: I should like to thank Mr. Cullen for his note, and also to ask if there are any practical men present who can give us any information on the point he has raised?

Mr. J. P. Ward (*Member*): With regard to the coloured wrappers for gelatine, I think it is very seldom that a stick will come out of a hole as neat in appearance as Mr. Cullen's specimens on the table. The paper is always burst in ramming the gelatine home, while in case of a partial explosion, the paper of contiguous cartridges is blown to shreds. I think, however, it would be a great advantage to have the paper coloured red as it would make men careful—seeing red paper anywhere, they would look for and expect to find explosives. Besides, white paper, or old newspaper, is generally used for tamping cartridges. Personally, I should prefer the red wrapper, the bright red exhibited would be more distinctive than green amongst "blue" rock, and sufficiently distinctive in "free-milling" rock.

Mr. R. N. Kotze (*Government Mining Engineer*): In connection with this question of coloured wrappers another point appeals to me. It is not so much that explosives should be given coloured wrappers different from those now in use as that the various factories should distinguish their products in that way. When accidents happen in these mines, it is often very difficult to determine what particular dynamite was used, and from what factory it was drawn, especially in gassing accidents. If it could be ascertained from what particular factory the explosives were drawn which were concerned in accidents of various kinds, the knowledge would be useful. The provision of coloured wrappers would assist in gaining the necessary information. I think this is a point which might be considered by Mr. Cullen, and other people who manufacture dynamite.

DIAMETER OF EXPLOSIVES CARTRIDGES.

Mr. Wm. Cullen (*Past-President*): I have had it represented to me by practical miners that the $1\frac{1}{4}$ in. cartridge, at present supplied by all manufacturers, is too large for the bottom of the hole. They say in effect that what they want is a 1 in. diameter cartridge for the bottom, because the ordinary size seldom reaches the bottom, its diameter being too large, consequently a certain amount of drilling work is thrown away. What they ask is, that every 50 lb. case should contain a certain number of 1 in. plugs. I have made enquiries in several quarters, both official and private, but the answers received are so contradictory that I am left in very considerable doubt as to what is the best course to follow. I should like, therefore, to have the point discussed by the members either in the form of correspondence or at the usual meetings.

Mr. J. P. Ward (*Member*): With regard to the idea of including a few 1-in. sticks of gelatine in each packet of $1\frac{1}{4}$ in. gelatine, I do not think this very necessary, as in the few cases where the $1\frac{1}{4}$ in. is too large for the bottom of the hole, the sticks may be very quickly unwrapped, reduced in diameter by rolling in the hands and rewrapped to suit. The men can tell by the fit of the charging stick in the hole when this is necessary. The inch sticks might be preferred by some men but it is so vitally important to fill the bottom of the holes tightly that I would not recommend them where $1\frac{1}{4}$ in. steel is used. On the mine I am now connected with, we formerly used 1 in. finishing steels in all our machine stopes, and 1 in. gelatine was procured to suit. It was found that in every contract, and for a period of about six months, the consumption of gelatine per fathom broken was considerably greater than formerly. It was decided to revert to $1\frac{1}{4}$ in. steel and gelatine and immediately the consumption became normal. It was thought at the time that the excessive consumption was due to the men not correctly gauging their charges, but as the test ran for six months, this may not have been so much due to that, as to a wrong distribution of the explosive in the hole.

The President: Do I understand that the men find a difficulty in gauging it?

Mr. J. P. Ward: They find a difficulty in estimating the quantity of dynamite to put into each hole.

The President: I think the use of some wrapper on explosives which would show itself is decidedly an advantage. In the course of my experience, I have found sticks of dynamite in all sorts of awkward places, and where you probably do not want to see them. Occasionally one finds a piece amongst the ore in the ore bins. You see a sausagy looking thing which bears a resemblance to the ore, and find that it is a cartridge of gelatine. If a coloured wrapper had been used a little bit of it would probably be found sticking on it, and might have called attention to it during previous operations. You sometimes find pieces on the sorting tables, passing the sorters and getting away down to the ore bins. With reference to Mr. Kotzé's remarks about the different manufacturers identifying their product by different coloured wrappers, that I think, is a good idea.

Mr. Lewis Evans (*Member*): I would like to say that the question of putting a number of inch cartridges in each packet of explosives has been considered by the Mine Manager's Association, and we decided against it. We think it preferable to have the holes made full bore at the bottom, because that is where we want the full power of the explosive.

FUSED SILICA WARE.

Mr. W. Cullen: At our last meeting Prof. Stanley referred to the use of certain forms of apparatus made from silica, and I gather from what he said that he was not aware that silica thimbles are made by the same two firms which have done so much to develop this special business. I saw them at a Royal Society Conversazione in London about 18 months ago, and although I cannot remember the price, they were infinitely cheaper than platinum.

The President: I understand that these thimbles are for parting and are made out of fused silica.

IMPROVED GLASSES FOR TESTING FOR POTASSIUM.

Dr. J. Moir (*Vice-President*): As you know, the usual test for K is done by examining the flame colour which the substance gives when held in the bunsen, through cobalt-blue glasses or an "indigo-prism." The principle underlying this is that these media cut off the yellow sodium light, but transmit the red potassium light. The great drawback of this simple test is that it is sometimes indecisive, for example, lithium, lead, arsenic, copper, etc., give flames which can be mistaken for the potassium flame when viewed through blue glass.

I have invented three different kinds of colour screens for this purpose, all of which are improvements on blue glass, and one of which is absolutely decisive even when lithium is present. They are made by staining gelatine films (old photographic plates "fixed" and washed), with various organic dyes, chosen so that the absorption bands of their spectra overlap, and therefore transmit only the extreme red (together with other colours, such as blue, which do not interfere). The films are then dried and cemented together.

The deep purple one is a combination of three films dyed respectively with fuchsine, methyl-violet, and haematoxyline. It transmits the extreme red down to C of the solar spectum, also a little blue round the F line. The greenish-blue screen is made of two films, one deep methylene blue (which cuts off lithium light) and the other methylorange. This transmits light from A to B and from E to F only. The third screen is single but is rather difficult to make; a film is dyed in very strong methyl-violet and then carefully dipped in picric acid until it transmits only the light from A to C. This screen is of no use if lithium is present; in fact it is quite similar to the deepest ruby-glass of the photographer.

The greenish-blue screen is very near theoretical perfection because, (1) the only red flame which it can transmit is that of potassium, and, (2) the green light which it transmits is not given off by any common substance, and, (3) the red K flame is seen in the midst of a contrasting background of greenish-blue due to the carbon monoxide of the bunsen flame and is therefore unmistakeable.

The principle of these glasses might be extended to the detection of other elements which give definite spectrum lines; it would be quite easy, for example, to invent combinations of dyes which would transmit only lithium or thallium light.

The President: I am sure we are very much indebted to Dr. Moir for his interesting little note, and I wish to thank him for bringing the matter before us. I think this is another practical proof of the advantage of holding our meetings in this theatre, where we have facilities for such demonstrations as Dr. Moir has given us.

ALTERATION OF RULES.

The President: You will notice that on the back of the agenda paper, there is a notice of a special general meeting, to ask you to sanction the alteration of a rule in regard to student members. It has been thought by the Council that it is a pity we cannot extend our student membership in some way, and not confine it only to the students of the University Colleges. We want to give young apprentices and young men who have lately taken up cyanide and battery work, and to some extent mining, an opportunity of receiving our *Journal*, and of coming here and hearing our papers read, and if they feel inclined, to take a part in the discussion. Therefore, I need not read this proposed alteration. You will see that it covers the admission of apprentices or learners as well as *bona fide* students of a Technical College.

Mr. A. Richardson (*Member of Council*) read the following paper :—

THE PILGRIM'S REST GOLD FIELDS AND MINING METHODS.

By J. Moyle-Phillips (Member).

Geography.—The township of Pilgrim's Rest—the centre of the Pilgrim's Rest Gold Fields—is situated in the district of Lydenburg, in the Eastern Transvaal, and 30 miles N.E. of the town of that name. It is distant about 80 miles from Machadodorp, on the Delagoa Bay railway, from whence there is a tri-weekly mail and passenger service by coach. Heavy transport from Machadodorp and Pilgrim's Rest is effected by means of mule and ox wagon, the transport rate at present ruling being 4s. 6d. per 100 lbs. This high rate of transport is in a great measure accountable for the apparent by high mining costs, especially when compared to the mining costs on the Rand. Recently a mail and passenger service has been inauguarated between Nelspruit, on the Delagoa Bay line, and the Sabie Gold Fields, 13 miles south of Pilgrim's Rest, which it is eventually intended to extend to Pilgrim's Rest township. The Nelspruit route is, without doubt, the route par excellence for Pilgrim's Rest. Being only sixty miles distant, as against eighty miles to Machadodorp, it is not only the shortest route but runs through the centre of the Sabie Gold Fields and thus taps the two richest and largest mining camps of the district.

Ten miles N.E. of the township of Pilgrim's Rest is the eastern escarpment of the Drakensberg and from the edge of the Berg a splendid panoramic view of the "low" country can be had, extending—on a clear day—to the distant Lebombo mountains. In some places there is almost a sheer drop of 1,500 feet from the top of the Berg to the bottom. To the S.W. of Pilgrim's Rest and from eight to sixteen miles distant are Mount Anderson, 7,489 feet above sea level, and Mauch Berg, 7,340 feet high, two of the highest peaks in the Transvaal.

The Blyde river, a tributary of the Oliphant's river, has its source about ten miles S. of Pilgrim's Rest and it is from this river that the mines obtain their main supply of water power.

History.—The Gold Fields of Pilgrim's Rest were first worked in the early '70's as alluvial diggings. Gold to the estimated value of £10,000, at 75s. per oz., was obtained in '70, whilst up to '77, diggers took out gold to the value of £500,000. It has been estimated that close on 2,000 diggers were at one time engaged in and around Pilgrim's Rest, but at present, owing to the scarcity of water, which has decreased 38 per cent. during the last ten years, no digging is being carried out. The famous Pilgrim's Creek, from which the bulk of the gold was won, runs between the hills on which the Jubilee and Theta Mines are situated, whilst the township is situated on both banks of the creek, about a mile above its junction with the Blyde river. Reef mining was started about '76 and the first mine—5 stamps—was erected at Stanley Bush, six miles S. of the township.

Geology.—The auriferous beds of the Pilgrim's Rest district lie mostly in the shales of the Pretoria Series and in the Dolomites, and occur in the following descending order:—

1. Languages Reef, in shales.
2. The Shale Reef, in shales.
3. Bevets Reef, at the base of the Pretoria Series.
4. Theta Reef, roughly 100 feet below the base of the Pretoria Series.
5. Beta Reef, 300 feet below the base of the Pretoria Series.

Of these, the only ones which are being worked profitably at the present time are the Theta and Beta Reef by the Transvaal Gold Mining Estates, Limited.

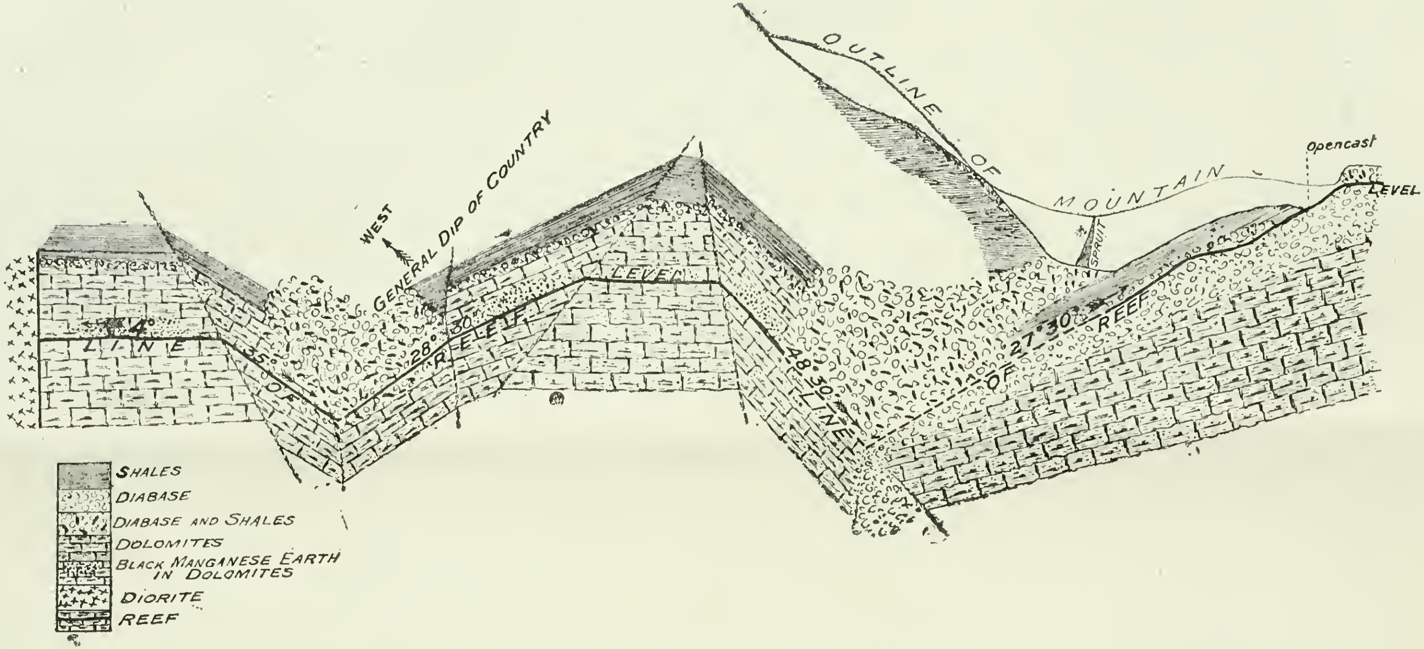
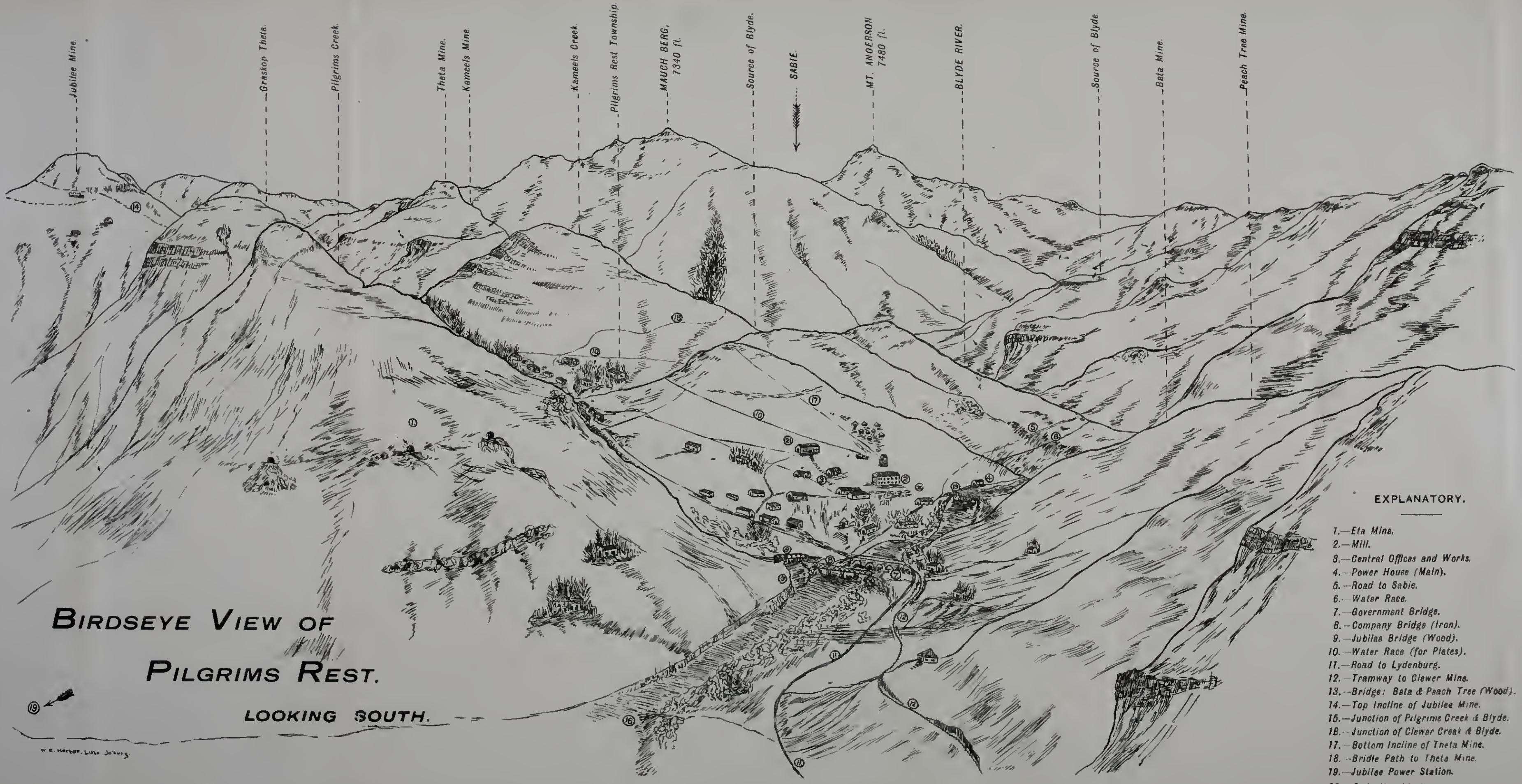


FIG. I.—Vertical Section along strike; scale 2,000 feet to 1 foot.

This section is taken along the strike East to West and shows how the strata has been disturbed. It will be noticed that the reef follows the undulations of the strata although it is evident that it was formed *before* the disturbances took place, as all dykes encountered penetrate through the reef and appear on the surface. There are several theories discussed here regarding the formation of this reef as to whether it was formed by infiltration, sublimation, or by the two, and also some people contend that there are not three separate reefs (Bevets, Theta, and Beta) but that the so-called Theta and Beta reefs are one and the same reef. It is said that the Theta and Beta reefs have never been found separately in one locality. On the Beta and Kameels Mine there is only one reef—Beta—whilst on the Jubilee and Theta Mines there is only the so-called Theta reef. Above the Beta Mine and on the contact with the shales is a reef corresponding to the Bevets, but there is no trace of the Theta Reef between it and the Beta Reef below. On the Clewer Mine again there is every indication of the three reefs—so-called—having come together, as in places the roof of the reef is the shales and there are distinct partings in the reef. The top portion is invariably the richest but very good values have been obtained from the bottom portions. On the Frankfort Mine there is a reef next below Bevets which the mining men of this district have not yet "placed" to their mutual satisfaction, some maintaining it to be the Theta, others maintaining it to be the Beta.

in
d-
ld

ne
y
ne
y
nt
ne
n-
ne
o-
e-
e
is
o
et
r-
al
g



BIRDSEYE VIEW OF PILGRIMS REST.

LOOKING SOUTH.

W. E. Hortor. Litho Jo'burg.

EXPLANATORY.

1.—Eta Mine.
2.—Mill.
3.—Central Offices and Works.
4.—Power House (Main).
5.—Road to Sabie.
6.—Water Race.
7.—Government Bridge.
8.—Company Bridge (Iron).
9.—Jubilee Bridge (Wood).
10.—Water Race (for Plates).
11.—Road to Lydenburg.
12.—Tramway to Clewer Mine.
13.—Bridge: Bata & Peach Tree (Wood).
14.—Top Incline of Jubilee Mine.
15.—Junction of Pilgrims Creek & Blyde.
16.—Junction of Clewer Creek & Blyde.
17.—Bottom Incline of Theta Mine.
18.—Bridle Path to Theta Mine.
19.—Jubilee Power Station.
20.—Reduction Works.
21.—Mens' Quarters & Hospital.

The Theta Reef may be termed a coalesence of two or three reefs, as, although in places it is met with intact, yet more often it is split up into two, and sometimes three parts, with a well defined parting between each. This reef runs from six inches to ten feet in thickness, but the average width may be taken as two feet. The Beta Reef is one well defined bed having an average thickness of ten inches, and stopes of only twelve inches in width have been worked in the Beta Mine. From two to thirty feet above the Theta Reef is a quartz leader from three to six inches thick, which is known as the "indicator." Immediately overlying this is a sheet of diabase two to forty feet thick, and next above is chert one to eight feet thick, and then the shales.

A remarkable body of ore was discovered in the Duke's Hill section of the Clewer Mine. It took the form of a gutter or channel with an average thickness of seven feet and increasing from fifteen to one hundred and fifty feet in width; after extending for some distance it suddenly terminated against a sharply defined fold of the shales.

The thickness of the strata between the Theta Reef and the diabase sheet is very variable, as in places the diabase forms the hanging wall of the reef whilst in others it may be forty feet distant, or, again, may be absent and the cherty dolomites or shales form the hanging wall. Pockets or masses of black manganese earth are frequently met with in the upper dolomites, and the conditions of development and stoping vary considerably in consequence. Wherever these earthy masses are encountered in developing, heavy timbering is necessary, whilst in the solid dolomites no timbering is required. Faults are frequently met with (Figs. I., II., and III.), and cracks in the formation varying from a mere fracture to several feet in width are often encountered, extending

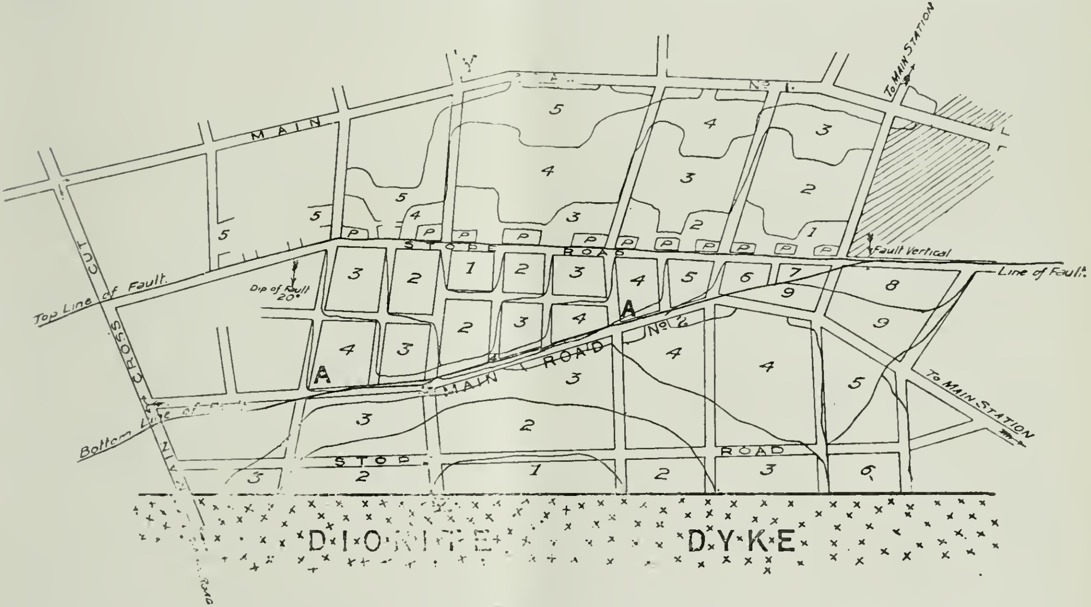


FIG. II.—Scale: 500 ft. to 1 foot.

From the main road No. 1 to the stope drives were put in until they reached the top of the fault. These were then connected by another small drive running on the line of the fault. Stoping was started by attacking one of the sections, from the stope drives on each side of it, ten feet away from the drive running on the fault. After the stope had been cut into for ten or twelve feet it was again connected with the fault drive thus leaving a pillar ten by twelve feet, a series of pillars were thus left along the fault. The stope was not worked out strictly on the long wall system as in coal mines but on the method shown, which gave a longer stope face and thus allowed of more hammers being employed at the same time. A pillar ten feet thick and the whole width of a section was left standing alongside the main road No. 1. The main road No. 2, and the drive parallel with the dyke, were driven simultaneously, the main road being driven along the bottom of the fault and the stope drive about ten feet away from the dyke: these were connected by other stope drives and the section thus blocked out. Stoping was done on the same method as adopted on the top of the fault with the exception that no pillars were left alongside of the dyke. In the meantime the section *on* the fault was being blocked out in the manner shown. When the bottom section had been stoped to within fifteen or twenty feet of main road No. 2, stoping operations were started on the fault. The tram line was broken at the point marked*, the pillar 1 was then taken out, and the remainder then attacked as numbered. The pillars left standing on the top section alongside the fault were taken out as they were reached and the whole section was thus stoped. Props were used to support the roof where deemed necessary, but 75 per cent. of these were afterwards withdrawn. From the point* the reef was trammed in both directions along the main drive to the main station. When the points A A were reached the roof collapsed but the faces were always at a safe distance from the fall. The only pillars left standing alongside main road No. 2 were on the dyke side, where, in places, the reef had pinched to six inches or less.

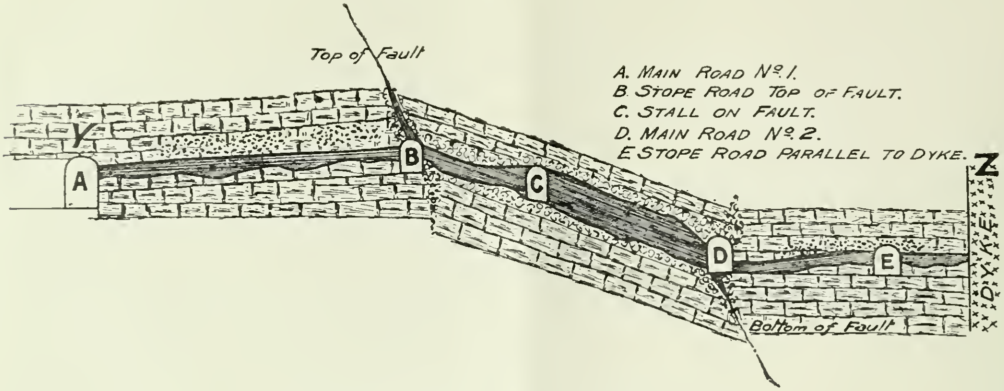


FIG. III.—Section from Y to Z. Scale 250 ft. to 1 foot.

A. Top main road. B. Stope drive along top of fault. C. Gallery.
D. Bottom main road, along bottom of fault. E. Stope drive parallel to dyke and 10 ft. from same.

The shaded portion indicates *Reef*. It will be noticed that the fault was entirely filled in with reef matter which was all payable and which opened out to eight feet thick in places.

both up and down through the strata to unknown distances and in places opening out into huge caverns. There are several dykes, mostly diorite, traversing the formation and one on the Beta Mine measures one hundred feet across, whilst others are from ten to fifty feet in thickness. The reefs themselves are composed of loose white opaque quartz embedded in, or surrounded by, a red or reddish brown earthy matter, probably the result of the decomposition of the pyrites which the reefs originally contained and which has left the quartz with a cellular or honeycombed appearance.

The gold in the oxidised portions is found "free," but there is an absence of "visible," although the values may run into ounces per ton: in fact the quartz itself carries very little gold, the bulk of it being in the loose earthy matter.

When the reef is highly pyritic the ore is very refractory, necessitating modification in the treatment at the reduction works. When the reefs are below twelve inches in thickness they are generally solid and require blasting, but above that width they can be generally worked out by picking. Around the outcrops and for considerble distances on the dip the reefs are oxidised, but as they get deeper under the strata they become pyritic. The dip of the strata averages 7° W., but in the vicinity of the mines it is only 3° 45′.

The mines of the Transvaal Gold Mining Estates, Ltd., are situated within a radius of 16,000 feet from the Central Mine and the lengths of their respective tram lines are:—

Clewer, 5,000 yds.	Peach Tree, 2,735 yds.
Jubilee, 5,560 yds.	Theta, 2,780 yds.

The following table shows the principal mines and the reef worked, also the height above sea level.

Company.	Mine.	Reef.	Feet above sea level.
	Frankfort	Shale Bevets Theta or Beta supposed	
Gras Kop	Gras Kop		
Theta ...	Theta ...	Theta ...	5,200
Transvaal Gold Mining Estates Ltd.	Clewer ...	Theta ...	4,387
	Peach Tree	Theta ...	4,310
	Beta ...	Beta ...	4,220
	Theta ...	Theta ...	5,100
	Jubilee ...	Theta ...	5,525
	Kamel or Chi ...	Beta ...	5,185

Development.—Owing to the irregularities in the formation, and the numerous changes in the levels of the reefs, no uniform system of development can be strictly adhered to. The adits cannot always be driven at equal distances from each other but have to be put in at those points which, with due regard to economic haulage, are deemed best, but, whenever possible, the main drives are carried from 250 to 300 feet apart and the main crosscuts 500 feet apart. These blocks are again intersected by stope roads 50 feet apart. Main drives and crosscuts are driven 5 ft. x 6 ft. high, and when timbering is necessary the sets are cut with 3 ft. caps, bottom spread 5 ft., and 6 ft. vertical height. In the solid dolomites no timbering is required, but where the black manganese earth is met with or the reef leaves the dolomites and comes in contact with the diabase, chert, or shales, heavy timbering is generally required. As showing the variable conditions of driving, the same drive may cost anywhere from 5s. to 45s. per foot, timbering included, but not the cost of the timber. The rate of development is from 40 feet in dolomites to 130 feet in shales per month, working double shifts. Each ganger on development has charge of from four to six faces which he has to keep timbered, if necessary, and has also to superintend his own tramming. All drilling is by hand labour, the average depth of hole in dolomites, per boy per shift, being 36 inches. Drilling is easy in the reef, but a boy has hard work to drill 36 inches in dolomite.

Stoping.—As no uniform method of development can be adhered to, neither, for the same reasons, can any particular system of stoping be adopted.

The size of stope roads depends on several conditions.

(1) If the nature of the surrounding strata allows, without undue timbering, the large mining trucks of 16 cubic feet capacity are led directly into the stopes and filled at the working face, but

(2) where this is not feasible, a smaller truck, 6 cubic feet, is utilized. This is filled at the working face and then either dumped alongside the main drive, to be again shovelled into the big trucks, or tipped direct into them—if the level of the reef is sufficiently high above the main roads to allow of this being done.

(3) When the extraction of the reef alone, does not give sufficient height for the passage of the stope trucks, the bottom of the drive is taken up to give the requisite height.

The general mode of stoping is to cut up the blocks by a system of stope drives and then commencing from the centre of the block, to extract the reef either by the "long wall" system or, when the ground is too heavy or loose, by the pillar and stall system. The latter method entails the splitting up of the small stoping blocks by means of galleries. When the block is thoroughly intersected in this manner the pillars are removed, commencing from the centre, and the roof allowed to fall in, but wherever practicable the "gobs" are filled in with the waste rock from development. When the reef is tilted at an acute angle as in Figs. 1 and 2 the overhand and underhand methods are resorted to, and thus in one particular section of the mine one may find all the different methods mentioned, in force, and this may even apply to one particular block in any section, such being the peculiarities of the formation in this district. However small the reef is, down to six inches in fact, it is taken out clean, that is, none of surrounding strata is broken with it, or, in mining parlance it is "resued." It may seem incredible, yet it is a fact, that stopes of only twelve inches in height have been worked in these mines, of course this has only been in those parts where the foot and hanging walls are solid.

These 12 in. stopes are still worked mainly by "umfaans." The ore is removed from the face to the stope roads on a sheet of iron roughly 2 ft. 6 in. by 2 ft. and slightly concave ; a rope is attached to each end of this iron sheet, one end being held by the boy at the face and the other by a boy in the stope road, and thus the sheet is dragged to and fro along the floor of the stope. The gobs are kept open when required by means of props, or cogs, and chocks of mining poles or waste rock. Holes drilled in stopes are seldom more than eighteen inches deep, if drilled deeper they are very liable to excavate at the bottom instead of bringing their burden. The charge seldom consists of more than one cartridge, sometimes only half a one, of gelignite, this being sufficient to loosen the reef and allow of its being picked out. Stope gangers have two or more blocks to over see and have to put in all props, cogs, etc., and timber stope roads—if necessary, also "boss up" their own trammers. Each stope gang consists of from thirty to fifty boys, including hammer, tramway, timber, drill carriers, and boss boys.

Timbering.—The timber used in the mines of this district is the ordinary bush timber, such as is found growing in the kloofs on Government lands and private farms. A contract for the supply of mining poles is let by tender yearly and is for cutting and transporting poles 7 ft. long x from 4 in. to 9 in. in diameter. Poles exceeding these dimensions are called "specials" and are paid for accordingly. The majority of the poles come from below the Berg, from whence they are hauled to the top on aerial gears by means of steam hoists or oxen ; from here they are transported to the different mines by mule or ox transport, the price, delivered on the mine, being:—9 inch 4s. 6d. each, 8 inch 4s. 2d., 7 inch 2s. 6d., 6 inch 2s. 2d., 5 inch 1s. 1d., and 4 inch 1s. The poles last from one to three years, according to the condition they are in when first used and the work they are utilized for, but they may have to be replaced much more often if subject to undue pressure or other adverse conditions. In the main roads it has been found necessary to replace them every two years.

As an experiment, poles of the black wattle were tried for sets in the main drives, and some of these have been standing for five years, but are showing signs of decay and will have to be replaced at an early date; but so far they have proved to be the best mining poles obtained locally. Blue gums have been tried but are found to be practically useless as props or legs of sets if subject to any great pressure, as they sheer their entire length in a very short time. If put in green they serve as a makeshift, but when once sun cracks have developed they are useless. There are two styles of notching used in framing the sets, which are locally known as the "Welsh" and "Cornish" notches respectively. As can be seen from Figures 4 and 5, the "Cornish" notch is cut square whilst the "Welsh" is bevelled. The "Cornish" has been found to be the best suited to withstand the movements of the strata, and consequently is superseding the "Welsh." As the greatest pressure on the sets, &c., is from the top, the notches in the legs of a set to fit the cap piece are cut in the butt end of the pole, thus the pole—as the leg of a set—is erected in the opposite position to that in which it grew, or, in other words, is put in upside down. The first parts of a set to show signs of weakening are the notches.

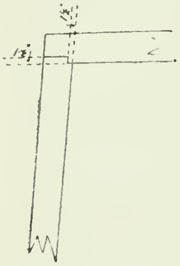


FIG. IV.—Cornish Notch.

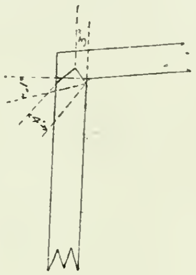


FIG. V.—Welsh Notch.

Poles from six inches to nine inches are used for sets in the main drives and crosscuts, those from five inches to seven inches for the auxiliary crosscuts and stope drives, and the four inches and five inches for lagging; for props any of these sizes are utilized as occasion demands.

The upkeep of the timbering in the mines is a very large item in the working costs, but this again is somewhat counterbalanced by the low consumption of explosives as compared with the Rand mines.

Haulage.—The underground haulage is accomplished by means of boys, mules, and electric hoists. Boys push the trucks along the easy gradients, but where more than two boys are required a mule is used.

Unless the working faces are near the outcrop—in which case the trucks are taken out direct—the reef is trammed to a central station, from whence it is hauled in sets of three or more trucks to the surface by mules or electric hoists. The type of mining truck used is the V-shaped side-tipping of 16 or 11 cubic feet capacity, whilst, as already stated, a truck of 6 cubic feet is used in the stopes. After leaving the mine, the ore is dumped into a big bin, from whence it is taken to the mill in sets of 11 to 14 trucks of 30 cwt. capacity each.

On the Clewer Mine an aerial gear is employed to convey the ore from the top levels to the main bin, but self-acting inclines are more frequently used. The full sets are run to the mill by gravitation or electric motors. From the Clewer Mine the sets, after being started by the motor, are run by gravitation to the junction with the Jubilee Mine (4,000 yards). The speed is regulated by brakes on the trucks and by the motor. From the Jubilee Mine the ore leaves the mine bins in sets of six trucks of 16 cubic feet by means of two self-acting inclines, 660 and 500 yards long respectively, to a station on the top of the main incline. From this point another self-acting incline lowers it to the bins at the foot of the mountain, 1,100 yards distant, of an average gradient of 1 in 5. From the foot of the mountain it is taken in sets of 10 to the junction with the Clewer Mine, 2,800 yards distant, by gravitation, and from the junction the trucks are hauled by motor to the mill, a distance of 1,000 yards.

From the Theta bins the trucks—in sets of 6 of 16 cubic feet—first pass down a self-acting incline 350 yards long, and are then taken along the contour of the mountain (2,000 yards) by mules to a point immediately overlooking the crusher house at the mill, from whence they are lowered by another self acting incline 430 yards long.

The ore from the Peach Tree Mine is first conveyed down a self-acting incline—155 yards—to the main bins, from whence it is run in sets of 14 trucks of 30 cwt. to the mill by gravitation, 2,580 yards.

The trucks from the Beta Mine, after travelling 100 yards by gravitation, join the Peach Tree line. From the main station at the mill the trucks are hauled up an incline by electric hoist to the crusher house.

Pumping.—There is only one pump in use in the mines of the Tranvaal Gold Mining Estates, Limited. This is at the Clewer Mine. It is a centrifugal with a 3 inch delivery and is electrically driven. Working in spells of 20 minutes duration four times per 24 hours it successfully copes with the water.

Whenever the water is met with elsewhere it is conducted to some fissure in the strata down which it disappears.

In the dry season the mines are comparatively dry, but when the rains are on, the water percolates through the soil and finds its way by means of the fissures, &c., into the mine.

Employees.—The white population on the mines is a very cosmopolitan one, although among the mechanics and staff the Scotch predominate, whilst among the miners the Welsh are in the majority.

The wages ruling are:—

Mechanics	20s. per shift.
Electrical Shiftmen ...	£25 per month.
Stopers ...	from 15s. to 20s. per shift.
Developers and Timbermen	20s. ,,

In the mine the day shift is from 7 a.m. to 6 p.m. and the night shift from 7 p.m. to 6 a.m., each shift being allowed an hour off for meals.

The white employees are provided with free quarters. At the central works there is a boarding-house for the convenience of the mechanics and mine men, but on the mines the men generally form messes among themselves and employ Kafir cooks. The cost of running a mess ranges from £4 to £6 per month, whilst at the boarding-house £8 is charged.

Owing to the mines being so scattered, there is no recreation hall as on the Rand, but there is a Sports Club which embraces every kind of game, and of which nearly every employee is a member. The average cost of living—outside of food—is much higher than on the Rand, owing, no doubt, to the high rates of transport. Each employee contributes 10s. per month for medical attendance, medicine and hospital. A Medical Officer is in attendance for both white and coloured employees, and there is also a hospital with a full equipment of medical and surgical appliances, including an X-ray apparatus.

Natives.—The natives employed are mostly local boys and Basutos, and, whilst the supply at present is about equal to the demand, it is very erratic, four months being the average length of a boy's contract.

The wages per 30 day month are:—

Boss Timber Boys ...	from	50s. 0d.	to 60s. 0d.
Drill and Pick Boys ...	,,	42s. 6d.	,, 50s. 0d.
Trammer Boys . .	,,	30s. 0d.	,, 45s. 0d.
Picanninis (drill carriers)	,,	7s. 6d.	,, 20s. 0d.

There is the usual allowance of meal and fresh meat per boy.

Wood and iron brick-lined compounds were erected for native accommodation, but they prefer living in grass huts as at their kraals, consequently the land around the mines is dotted with those huts, around which is generally a mealie patch planted by the boys for green mealies.

For cooking purposes the boys are squadded into gangs of from 16 to 25 with a cook to each gang, the food being cooked in the ordinary 3-legged Kafir pot. There being numerous kraals in the vicinity of the mines, the week ends are generally devoted to "tywala" drinking, which is brought in by the wives and sisters of the boys working, and as the mine officials keep the supply within reasonable limits there is no loss of time through drunkenness, and the boys appear to be quite content.

General.—Around Pilgrim's Rest are several small mining propositions which are worthy of note.

The Grasskop Theta Syndicate is working claims 1½ miles S.E. of the township on the hills opposite the Jubilee Mine, whilst adjoining ar several smaller concerns held on tribute and in course of development which are turning out well. N.E., and about 15 miles down the Blyde river, is the Vaalhoek Mine (Transvaal Gold Mining Estates, Limited), on which it is intended to erect a 10-stamp mill, whilst adjoining it the Grasskop Theta is working some claims. These are situated on the Glynn's Horizon—below the Beta Reef. Nearer home and more to the N. is the Frankfort Company. 18 miles S. of Pilgrims' Rest are the Glynn's Lydenburg and the Glynn's Extension Companies, working the Glynn's Reef. Quite recently a new find has been made close to the Sabi Falls in the sandstones and about two miles N. of Glynn's Reef; several small mines are at work here and doing well; these are in the hands of private individuals. To the E. of Pilgrims' Rest is the defunct—as far as working is concerned—Lisbon Berlyn Co., and also the farms London, Kimberley, Belvedere, and several others, on which are very rich leaders from 1 inch to 6 inches thick but very patchy: several of these have been and are being worked. 25 miles W. are the Finsburg and Nooitgedacht mines on the Finsburg, Button and Davidson Reefs, lying in the shales: these are being worked on tribute to the Transvaal Gold Mining Estates, Limited.

In and around Pilgrim's Rest are numerous small propositions worthy the attention of the prospector and small capitalist. The reefs are generally easily worked, and the power (water) is ample for anything up to a 5-stamp proposition. As these small mines are likely to attract more attention in the future, some notes on them may not be amiss. Undeveloped claims fetch about £20 to £25 each and developed claims about £100, the price depending on assay values and economic conditions. Leases are generally let for three years, with an option of another two years, and at 10 per cent. on actual gold won. Claim licences amount to 2s. 6d. per claim on Government ground and 5s. per claim on private ground.

The following was the price paid for a 5-stamp mill with a portable engine :—

Cost in Johannesburg ...	£375
Railage	£30
Transport to mine	£60
	£465

The stamps were 750 lbs. weight each. This price does not include any cyanide plant or any erection costs.

The gas engine is highly thought of where water power is not available, and under these conditions the 1-stamp gas mill seems but suited for these fields; the duty is high, up to 25 tons per 24 hours, whilst the cost of fuel, 2s. 6d. a bag for charcoal, is low. A Tangye suction gas engine driving 10 stamps, crusher, and two centrifugal pumps at Barberton, consumed 3½ to 4 bags of coal per 24 hours, equivalent to 1 lb. coal per horse power per hour; when using charcoal the consumption was 7 to 8 bags per 24 hours.

Charcoal has the advantage of leaving no tar and no clinkers are formed in the gas generator. Another Tangye suction gas engine plant belonging to the Messina Development Co., Ltd., rated at 116 B.H.P., but only loaded to 40 or 50 B.H.P., ran during one month 30 days 14 hours, including a non-stop run of 21 days 6 hours The fuel consumption amounted to 443 bags of local charcoal of 65 lbs. weight each and costing 2s. a bag, equal to £44 6s. 14 per cent. of dust, fines, and waste was sieved out, leaving 380 bags actual consumption. On the basis of 443 bags, the consumption was 0·84 lbs. of charcoal per B.H.P., per hour, or a cost of 0·36d. per B.H.P., per hour, or, on a basis of 1,300 tons, 1s. 6d. per ton crushed.

A scheme is on foot to erect an electrical supply plant, guaranteed by the Government, in the vicinity of Pilgrim's Rest, and it is estimated that a 5-stamp proposition can be run by this means at a cost of £30 a month.

The President: I would like to move a hearty vote of thanks to Mr. Phillips for his interesting paper. I think this is the first description we have really had of the Pilgrims' Rest goldfields, and we have had the advantage of having the paper read by one who knows the district, and is able to lay stress on various little points which might otherwise have escaped notice.

The vote of thanks was agreed to.

Mr. W. A. Caldecott (*Past-President*): I should like to offer a comment upon one point in this paper with which the author does not deal very fully. He mentions the gold in the oxidised portion of the ore as being found free, and that nevertheless there is an absence of "visible." In my experience of the ores of that district a good many years ago I also observed this fact, and I think that it is due to the extremely fine state of division of the gold so that when the ore is panned the gold floats in the water without sinking to the bottom. As an illustration of this, I may say that on one occasion I came across a prospector who showed me, with a certain amount of pride, the reef that he had struck. On asking him what he thought its gold contents per ton were, he estimated the value at 10 dwt. The assay value turned out to be 46 dwt., but the 10 dwt. estimate was probably not much, out so far as possible recovery by amalgamation only was concerned. In the days before the introduction of the secondary process of cyaniding, and when exceedingly rich ore was treated in batteries and grinding pans in the Pilgrim's Rest district, the loss of fine float gold in the tailings flowing into the Blyde River must have been very high.

Mr. W. R. Dowling (*Member of Council*) read the following paper :—

NOTES ON PRECIPITATION.

By Mather Smith (Member).

The object of this paper is an attempt to prove, that the generally accepted principle that it is necessary to have one cub. ft. of zinc shavings, for the precipitation of one ton of solution in 24 hours is wrong: that more zinc than is necessary, is being generally used; that an excess of zinc, is both harmful and costly; that precipitation boxes, are often constructed on a wrong principle; and that a comparison of the rate of flow of solution per cubic foot of zinc, which does not also include or consider the difference in area and number of compartments in the precipitation boxes, is misleading. If one cub. ft. of zinc is required for the precipitation of one ton of solution in 24 hours, then, by simple multiplication, 2 cub. ft. of zinc will be necessary for the precipitation of 2 tons of solution, in 24 hours, and this is the rule which appears to have been acted upon by those responsible for the construction of the boxes on some of our latest cyanide plants.

But there are four different possible ways of arranging these two cub. ft. of zinc, as they may be either in series, vertically or horizontally, or in parallel, separately or together. Although in each arrangement the total flow of solution would be the same: with those arranged in series it would flow through each cubic foot of zinc at the rate of two tons per 24 hours, whereas through those in parallel it would be at the rate of one ton per 24 hours only,* and does it not seem reasonable to expect that those arranged in parallel would give better precipitation than those in series? At any rate, the danger of having gold slimes in suspension carried out of the boxes would, in those in parallel, be reduced to a minimum.

One group of mines started several years ago dividing their boxes of eight compartments, into two boxes of four compartments each, and are now installing, low wide boxes of four or five compartments each. If these are giving good results, why are boxes of seven and eight compartments still being constructed? If four or five compartments are sufficient: what is the necessity for the other two or three?

In my opinion, the superfluous zinc does a great deal of harm, as it must occasion an unnecessary consumption of cyanide, and charge the solutions with an excess of zinc, which, in circulating through the plant, will in a great measure, be deposited as white precipitate at the heads of the boxes. Many will disagree with me in this, but I have personally observed, on two different cyanide plants, on which the sands and slimes solutions were at first run separately, that when the weak solution from the sands plant was used on the slimes plant, there was an immediate increase in the amount of white precipitate in the slimes boxes which I could only attribute to the zinc from the sands solutions. May it not also be possible, that solutions which are comparatively free from zinc, will allow of better precipitation than those which are heavily charged?

Another thing which must still further reduce the area required for the free passage of the solution, is the way in which, on many plants, the zinc is pressed down into the boxes. It is surely unnecessary in an electrical process of precipitation, to endeavour to get each and every molecule of the solution into actual contact with the zinc. The solution gradually forces open channels for its free passage, and I have frequently seen, on more than one plant, the solution rising up through one or more circular holes in the mixture of zinc and white precipitate in the top compartments of the boxes, and making small geysers on the surface. The same thing must be taking place, although not visibly, in the other compartments, and must obviously tend to carry off gold in suspension.

I had long held the opinion that more zinc than was necessary was being generally used, and on taking over the management of the cyanide works on the Worcester G. M. Co., Barberton, at the end of April last; I started putting my theories into practice. There were here four precipitation boxes, three for the sands solution, and one for the slimes. The sands boxes had each six compartments full of zinc, and the slimes box had seven. Each compartment measured 36 in. wide, 22 in across, and 24 in. deep to the top of the sieve and all were packed as full as possible with zinc. I took samples of the solution sumps, and had them assayed with the following results:—

	dwt.
Strong sump ...	·6
Medium sump ...	·225
Weak sump ...	·225

I cannot find the record of the slimes sump assay, but it was also very high in gold.

I am not trying to prove that the large amount of zinc in the boxes had anything to do with the bad precipitation, but I am sure that many cyanide managers have experienced the same difficulty which I then had, and that was, to know what to do with the zinc in the boxes so that I could replace some of it with fresh zinc. There was no room for any at the heads of the boxes, and I had to run down a lot of it with sulphuric acid, in order to make room for fresh. Under my present system, there is always plenty of room at the heads of the boxes, so that should anything go wrong with the precipitation, and I have known that to happen on many plants, the boxes can be dressed up without any trouble.

To continue, each box, at the end of April, contained, on an under-estimate, 60 cub. ft. of zinc, and the solution during that month had averaged, in the sands boxes 47 tons per box per day, or ·78 tons solution per cubic foot of zinc per 24 hours. I started reducing the amount of zinc in the boxes, and now, instead of putting in 24 in. of zinc into the bottom compartments, when dressing a box, I put in 12 in. only. This afterwards settles down to about 9 in. It took about three months to gradually reduce the quantity of zinc to its present state, namely, five compartments in each sands box, with an average depth of zinc in each compartment of less than 9 in. As the sands tonnage treated has considerably increased since then, and the charges are also getting more solution, the sands boxes are now running at an average rate of 63 tons per box per 24 hours, or over 3 tons of solution per cubic foot of zinc per 24 hours. The weak box does more than the average. I had the slimes box made into two boxes, the one of four and the other of three compartments. The one contains, on an over-estimate, 16 cub. ft. of zinc, and the other 12. The usual flow of solution is at the rate of at least 65 tons per box per 24 hours, which works out at 4 tons of solution per cubic foot of zin per 24 hours, in the one box, and at 5·4 in the other.

---

* The rate of flow per sq. ft. of cross section being reduced by half.

I will now explain my method of dressing the boxes. When the strong or medium boxes require more zinc, the zinc from the top compartments of the slimes or weak boxes is lifted out and shifted over into them. The second compartment of the weak or slimes box is then lifted slowly up by the handles attached to the tray and put into the top compartment, tray and all, without any handling or disturbance of the zinc, and so on to the last compartment, which is dressed with new zinc.

This is usually done every fifth day, so that no zinc, in the one slimes box, is ever more than fifteen and in the other more than twenty days old. In the weak box, also, no zinc is ever more than twenty days old, as two compartments are frequently shifted out.

New zinc is put into the bottom compartments of the strong and medium boxes twice a month. No free cyanide drip is used, and the zinc, not having been disturbed, starts precipitating immediately, and, were it not for occasional traces, I might say that there is an entire absence of white precipitate.

The zinc in the boxes being shallow and loose, the gold slimes do not clog up the boxes, but fall to the bottom of the tray and leave the zinc clean and fresh.

My reason for continuing to fill four compartments with zinc in the one slimes box, when the other, with three compartments and running at the same rate, gives equally good results, I am unable to give, and can only attribute it to prejudice and years of training in the "more zinc better precipitation" theory.

I find that five compartments are necessary in the strong and medium boxes, as the zinc in these boxes, having been longer in use, appears to have lost some of its efficiency as a precipitant.

A sample of 100 c.c. of the solution leaving the slimes boxes is taken every 8 hours, and 20 A T.'s of the mixture are evaporated for assay at the end of each week, and since 9th August nothing higher than a trace has been returned from the assay office. Owing to the pressure of work in the assay office, the solutions from the sands boxes have not been sent up regularly, but so far, the highest assay from solution leaving the boxes has been one of ·01 dwt. from the strong box. The weak and medium boxes have been "nil" and "trace."

It has been suggested to me that the solutions here may allow of quicker precipitation than those on the Rand. In that case, why were the sumps here full of gold at the end of April last, although the boxes were full of zinc? It has also been suggested that my system of dressing the boxes would not be suitable on a large plant, as it would mean much extra work. I fail to see this, as it would simply require stronger trays and a proportionate increase in the number of assistants. I can dress my five boxes, with two boys to help me, in a little over an hour.

I attribute the good precipitation, to the freshness of the zinc, to the freedom of the solutions from excessive zinc in solution, to the freedom with which the solutions can pass up, without making channels through the zinc, and also to the fact that the hydrogen evolved has free egress and does not clog up the zinc with bubbles.

My actual extraction from the sands for the past three months, in spite of an increased tonnage treated, and a shorter treatment, has been 4% better than the average of the eighteen previous months of which I can find a record, and I strongly hold to the opinion, that the freedom of the solutions from zinc has a lot to do with the improved extraction. My experience here has convinced me, that if precipitation boxes were constructed to allow of one square foot, in area, for the precipitation of one ton of solution in twenty-four hours, with a maximum height of twenty-four inches from the top of the zinc frames to the top of the solution, and with a maximum of five compartments in each box, there would then be no possibility of any mistake being made in the construction of precipitation boxes.

I append the following table showing the consumption of zinc and acid, and the value of the gold slimes, for October and November 1908.

	Oct.	Nov.
Pounds zinc used per ton treated	·19	·25
Pounds zinc used per ounce fine gold ...	1·2	1·8
Pounds sulphuric acid used per ton treated ...	·09	·07
Pounds sulphuric acid used per ounce fine gold	·55	·5
Percentage fine gold in gold slimes ...	33·8	35·2

The zinc used includes waste, which, as we have a very poor lathe, amounts to a good deal

Since writing this paper, I have had two high assays from solutions leaving the boxes, the one of ·2 dwts. from the slimes boxes, and the other of ·075 dwts. from the strong, but as these were sent up during "clean-up," and the next assay from the slimes boxes was again "trace," I am sure that they were "salted." I have often had trouble here with salted samples taken at "clean-up," as the smelting room and extractor house are all in one.

The President: On behalf of the Society I beg to thank Mr. Mather Smith for his interesting paper.

NOTE ON A PROBLEM DURING SHAFT SINKING.

(Read at September Meeting, 1908.)

By Chas. B. Saner, M.I.M.M. (Member of Council).

REPLY TO DISCUSSION.

I regret that no members have seen fit to bring forward any problems that must have been met with in some of the shafts of the Rand during recent sinking operations; surely the work cannot have all been plain sailing.

At the risk of wearying you (and jumping in where wise men fear to tread), I will briefly describe a second and somewhat similar though more difficult and dangerous problem, that faced us in the Turf shaft, and our method of overcoming it.

For three months after settling the trouble described in my last note, excellent sinking was done, an average of 159 ft. for that period was maintained; then at 3,100 ft., or 540 ft. below, that bugbear water was struck in the north-east corner of the shaft, and then we sank 74 ft. only. At 3,140 ft. the formation became disturbed, and a fault appeared in the same corner, with a dip of 70°—80° south-west. At 3,170 ft., a narrow dyke with a dip of 73° south-west and parallel to the faults was cut, which had 6 in. to 18 in. of soft mud on the footwall, along which the water was flowing. The water now increased to some 30,000 gallons per 24 hours. No one has any idea what a drawback to sinking, even a small quantity of water, is when on the shaft bottom. At 3,195 ft., the position was as per Fig. I.

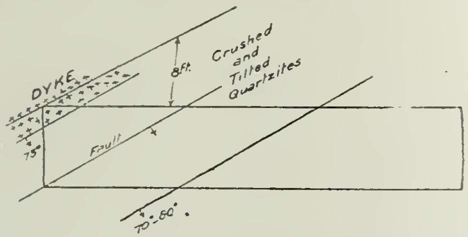


Fig. I.—At 3,195 ft.

The soft mud was being washed away by the water, thus undercutting the formation on the east side, which was already broken up by the faults. At this point the north-east side was breaking 8 ft. wider than necessary, the ground coming off to the dyke. At 3,171 ft., bearers were placed under the north end piece in good deep hitches, and at 3,177 ft., bearers were put under No. 1 divider, but after sinking further, the hitch on the east side became so weak, although cut in 6 ft. from the timbers, that it was found necessary to put in a vertical leg 30 ft. long, and 12 in. × 12 in., under the bearer, with the foot let into the solid quartzite through the dyke and cemented in. This leg was secured to the bearers with steel straps. At 3,190 ft., the bearers under No. 3 divider, where the wall plates are jointed (I may here mention that the wall plates are sent down in two halves, and butt-jointed, the No. 3 divider with studdle binding the joint), were also found to have a very insecure hitch on the east side, although cut in 12 ft. beyond the shaft timbers, and as the ground above was very heavy (because of the water washing out the decomposed dyke) and getting heavier every hour, and the formation below the hitch being broken, a diagonal bearer 30 in. deep was put in with a strong hitch on the west side and cemented. The wall plates were strapped to this diagonal and to the horizontal bearers, with steel angle iron, and heavy blocks were firmly wedged in as stiffeners, between the diagonal under No. 3 divider, vertical leg under No. 1 divider, and north end of shaft, so that any movement was impossible.

On referring to Fig. II., which is a vertical section north and south of the shaft, looking east, it will be seen how, as sinking was continued, the support or toe of the broken ground between the dyke and faults was being cut away and, with the water, making the east side very dangerous; thus when the shaft got to 3,226 ft. the problem became exceedingly ugly.

On the east side, "A to B" was the dangerous area, and the west side, "C D" was broken up, and tending to fall away from the dyke into the shaft, and between these points no solid ground could be found on which to place bearers; if solid on one side, it was weak on the other. All sinking was stopped, and the following method adopted, to overcome the difficulty.

Under the north end piece, good hitches were cut in each corner, and 36 in. bearers put in. Under the No. 3 divider, where the wall plate joint comes, a good solid hitch was cut on the east side, and a long 5 ft. hitch, which was not over good, on the west side, and 36 in. bearers put in. These bearers are east and west across the shaft. On these again were laid, on the west side from north end piece bearers to No. 3 divider bearers a beam 14 in. × 14 in. and 24 ft. long, and into this beam were dovetailed timbers, 16 in. deep, under north end piece and under No. 1

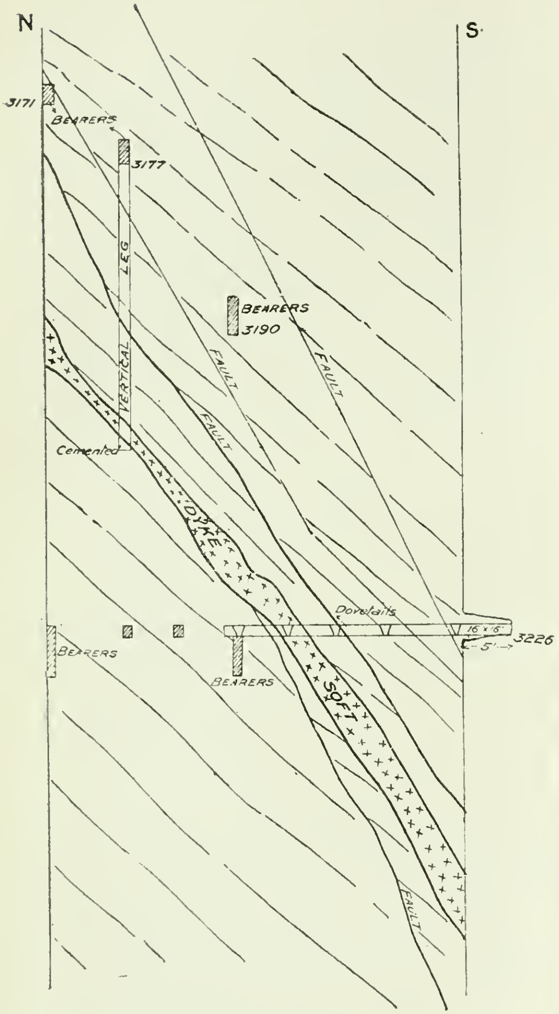


FIG. II.

2 and 3 dividers; each of these timbers was cemented into solid hitches on the east side. From No. 3 dividers into a hitch cut 5 ft. into the south-east corner, was placed a beam 16 in. × 16 in. and 28 ft. long, and into this beam was dovetailed 16 in. deep timbers under south end piece, and under Nos. 3, 4, 5 and 6 dividers, and each of these timbers was cemented into good hitches on the west side. (See Fig. III.)

Thus a solid framework was built in with supports, whenever sound ground could be formed, so framed that both vertical and side pressure would be resisted. A strong close bottom was put on this, the 3 ft. centre setts were close lagged, and the empty space behind, filled in with rock from the bottom, as previously described. Sinking was then continued with great care so as not to disturb the job with too

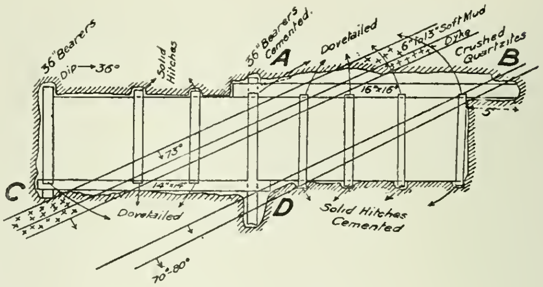


FIG. III.—At 3,220-30 ft.

heavy blasting. The ground on the west side on further sinking became bad, and the hitch under No. 3 divider bearers at 3,226 ft. became undercut because of the faults and dyke. (See Fig. IV.)

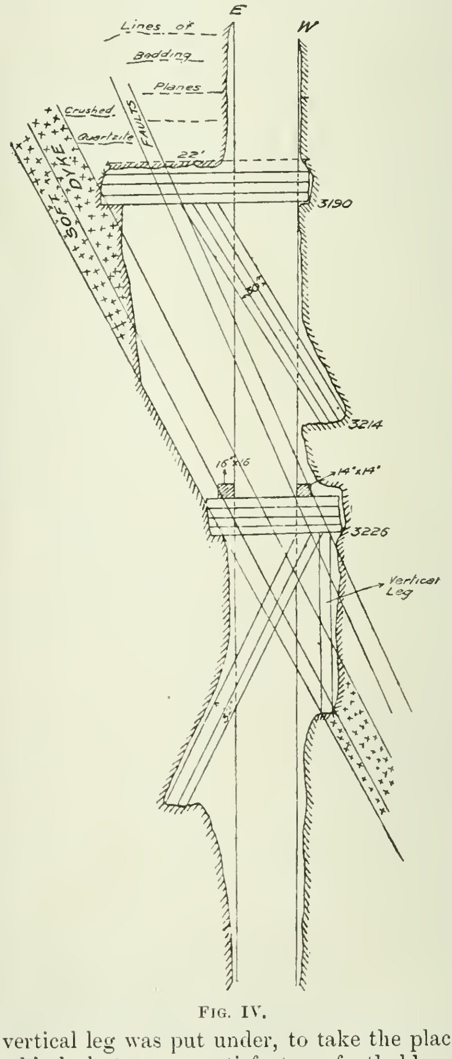


FIG. IV.

A vertical leg was put under, to take the place of the hitch, but as no satisfactory foothold could be got for this leg, a 32 in. deep diagonal was put in under the 3,226 ft. bearers on the west side, and into a good hitch on the east side. Thus the shaft was secured with a lattice work of diagonals through the disturbed area. At 3,294 ft., when once more in solid regular formation, a complete sett of heavy bearers was put in across the shaft as a final security.

I may mention that during sinking over 3,600 ft. in this shaft, seven distinct dykes have been passed through. When hard, but bringing in no water, progress is slow, the timber is damaged with blasting, but the sides are usually fairly good and ordinary care only is required. But when soft and wet dykes, either separately or both together, are met with, work is very dangerous and complicated, and I can say here with pride, that in spite of the several troubles encountered, no life, black or white, has been lost in overcoming them ; and this I attribute to the care exercised by all concerned.

The President : I am sure we must thank Mr. Saner very much for bringing these extra notes before us. He seems to have encountered great difficulties in sinking, and to have shown a great deal of ingenuity in overcoming them.

WHITE LABOUR IN MINING.

(*Read at January Meeting, 1909.*)

By Tom Johnson (Member).

DISCUSSION.

Mr. Ralph Stokes (*Member*) : The author pleads with such obvious sincerity for the employment of more white men underground, that it is unfortunate he does not support his case with a few more decisive facts in place of general assertions, highly theoretical estimates, and an appeal to sentiment. At times one may even suspect that he desires to involve the Society in a debate on social politics, and that his next theme will be women's suffrage, or the licensing bill. It is surely not within the province of this Society, wide though it be, to advocate the cause of white labour, save on purely economic grounds So far as the business side of his paper is concerned, the author leaves his first proposition—that more whites be employed on shovelling and tramming—in a state of such obscurity that it is fairly safe from criticism

The arguments advanced are, too shadowy for attack. One might as well go shooting protoplasm with an air gun However, the author does commit himself to the vague statement that the Chinese were better than the natives, but "did not get anywhere near" what he has "seen white men do." Such general opinions as these, do not help us to see how the whites are to be handled, and formed into a steady working force, to compete successfully underground with Kafirs, and at the same time earn a decent living wage. I hope he will touch upon this trivial question in his reply. When the smooth working of a "white" lashing and tramming system in other countries is cited, it is necessary to bear in mind, that this class receives as a rule, only about 30% less money than the competent machine man.

The author declares that we are too much in the hands of the natives, and that "stranger things than a strike amongst Kafirs have happened ;"—true enough, stranger things have happened, but a strike amongst white shovellers is not one of them.

The paper contains repeated references to how the unskilled whites on the Rand are "looked down upon," by the "aristocratic skilled men" who are said to forget, how short a time has elapsed since they themselves were "unskilled." His vivid picture of their contempt, is most astonishing. Few imagine the Rand miner to be such a snob. But the author's diagnosis is, I think, entirely at fault. The skilled man's antipathy is not a system of contempt, but is due on the contrary, to a full realisation of the narrow margin mentioned, and to the consequent dread of greater competition.

Turning to the second part of the paper, *re* "more whites on machines," it is satisfactory to find definite estimates given. For the purpose of his comparison, the author discreetly takes low mining efficiencies upon which to improve. For his representative case, he assumes an average fathomage per machine shift of ·5—the best men breaking ·8, and others down to ·3. As the average for five mines, for which I happen to have the figures, is ·7, the author seems to have a good deal of lee-way to make up before he reaches present standards under well run normal systems. We find him taking his ·8 men off their machines, by which they were probably making £2 a shift, and making them professors of mining to the lower ranks, at 19s. per shift. This, of course, would put the expert miners in a proper spirit of ardour, for the success of the scheme. The ten drillers supervised, would likewise welcome the minimised prospect of making big contract cheques, to which they might have risen with greater experience by putting their brains into their work, under the good influence of personal responsibility.

As an auxiliary and apparently not essential part of his scheme, the author advocates the introduction of a rigging and blasting gang The experiment has already been tried, I believe with promising results, and as those responsible, may perhaps favour the Society with their conclusions, the idea need not be theoretically dealt with. Taking the first proposal on its own basis, we may compare costs as follows:—

Under normal conditions, an average of ·7 of a fathom per machine shift, can be maintained under keen and helpful supervision. Let this be reduced to ·6. Then to break 74 fathoms, 123 machines are required, and a labour force of 41 miners and 287 natives. With gelatine at 16 lb. per fathom, costs appear as follows:—

41 miners at 30s. a shift ...	£61	10
287 natives at 3s. ...	43	0
Gelatine at 47s. per case ...	55	10
Fuse, detonators, etc. ...	7	0
Total ...	£167	0

Cost per fathom, 45s.

This figure is not merely an estimate, but practically coincides with the actual cost on five representative mines for six months.

Now, in the author's case, he first of all assumes the low average of ·5 fathom per machine shift. By taking off his ·8 men, this average is automatically reduced to about ·45, which he has to raise to ·7 by special supervision. Even allowing this unreasonable improvement of 55%, the possibility of which predicates an abnormally poor prior standard of mining and supervision, and also allowing the men to do their own charging and blasting, his costs compare most unfavourably.

74 fathoms would require 106 machines, with 106 white drillers, 10 bosses, and 141 boys (1⅓ per machine). Then

116 whites at 19s. ...	£110	0
141 natives at 3s. ...	21	0
Gelatine at 47s. per case ...	55	10
Fuse, detonators, etc. ...	7	0
Total ...	£193	10

Cost per fathom, 52s. 6d.

This means an increase of 17% in cost to the mine and a 33% decrease in average earnings to the men.

If we work out the two schemes on a more rational basis, giving the expert men their former earnings and granting, say, a 30% increase in efficiency, the loss, of course, is greatly increased. But the author's estimate is given the benefit of every doubt. Furthermore, it is assumed that suitable instructors can be found to effect the 55% increase in efficiency, and to benevolently give away the hard-won secrets of their trade to men who would become their competitors when the scheme collapsed. After a year or so of this costly tuition, the men would be individually well nigh as efficient as the boss, and then to give his policy support on grounds of economy, the author would have to manipulate his estimates with some ingenuity.

THE GASES RESULTING FROM THE USE OF HIGH EXPLOSIVES.

(Read at November Meeting, 1908.)

By Wm. Cullen (Past-President).

DISCUSSION.

Dr. J. Moir (*Vice-President*): I should like to call the author's attention to the fact that neither I nor he is the pioneer in showing that explosives give off CO on firing. In Sanford's[*] book I find (p. 142) an analysis of gelignite fumes (apparently obtained by firing in a bomb), giving 7% of CO along with 25% of CO_2 and 67% N_2. This analysis is sixteen years old, and is the first in which CO appears.

The President: I am sorry there is no discussion on so many of the papers before us. Some time ago we had a plethora of discussion and but few papers, now we have plenty of papers and no discussion. The main object of contributors in giving us these papers is to have discussion upon them. I think our members should take this to heart and see if they cannot manage to raise some discussion on these papers. It is disappointing to the authors, and the discussion is often far more valuable than the original paper.

The meeting then closed.

THE ESTIMATION OF CARBON MONOXIDE IN MINE GASES.

By E. H. Weiskopf, Ph.D., F.C.S. (Member).

Appendix.

Figures and data involved in analysis of gases given off by gelignite.

Determination of Amount of Sample taken.— The sample bottle after connecting with apparatus showed a temperature of 23·5° C. and 630.7 mill barometric reading; an internal pressure of 16·0 mill above the barometer. During the course of the analysis 12,209 gm. of mercury (S at 24·5° C. = 13·5350) equal to 902 c.c. had been introduced into the same bottle. At the end of the estimation the temperature had risen to 24·5° C., the barometric pressure remaining at 630·7 mill. The original volume of the sample bottle was decreased by 902 c.c., and therefore, provided that the remaining gas was brought to the same temperature and pressure conditions as existed originally in the sample bottle, this decrease corresponds to the volume of gas displaced and used. It therefore remained only, to bring the gas left in the bottle by means of the aspirator, to that pressure which at the changed temperature corresponds to the original condition. The formula is :—

---

[*] Analysis by W. J. Orsman, F.I.C., in 1893: quoted in Sanford's "Nitro-Explosives." (Crosby, Lockwood & Son.)

$$P_1 = P_2 \frac{273 + t_1}{273 + t}$$

In our determination, $P_1 = 648{\cdot}9$ mill ; and the pressure inside the bottle was consequently adjusted to that figure. The volume of gas taken from the sample bottle, now corresponds to the volume of mercury introduced, *i.e.*, 902 c.c. at 23·5° C. and 646·7 mill. pressure, which equals 707 c.c. N.T.P.

In cases where the gas samples are in a moist condition, the influence of the vapour tension due to the change in temperature must be allowed for, by adding or subtracting this difference to or from, the results obtained by the above formula.

ESTIMATION OF CO BY TITRATION OF IODINE SOLUTION WITH THIOSULPHATE OF SODIUM.

A freshly standardised solution of sodium thiosulphate was prepared for each analysis, containing about 5 gm. of this salt per litre. For the standardisation, freshly sublimed iodine was taken, starch being used as indicator for this, as well as the actual estimations.

·1674 gm. iodine dissolved in potassium iodide solution made up to 250 c.c.

20 c.c. of this solution equal to ·013392 gm. iodine required 48·90 c.c.

Hypo solution (mean of 3 titrations).

The iodine solutions from the analysis required :—

Before blasting	...	3·80 c.c.
After blasting	...	71·80 c.c.

From the equation

$$I_2O_5 + 5CO = I_2 + 5CO_2$$

1 gm. of iodine is liberated by ·551833 gm. or 441·4663 c.c. of CO.

Therefore we obtained :—

Before blast : 3·80 c.c. = 0·45337 c.c. = ·028 vol. per cent. CO.

After blast : 71·80 c.c. = 8·6803 c.c. = 1·228 vol. per cent. CO.

GAS-VOLUMETRIC ESTIMATION IN MODIFIED ORSAT APPARATUS.

Gas taken .	139·85 c.c.	at 21·5° C.	= 139·85 c.c.
After KOH absorption	131·85 c.c.	,, 21·4° C.	= 131·18 c.c.
After fuming H_2SO_4 ...	131·12 c.c.	,, 21·4° C.	= 131·17 c.c.
With palladium tube affixed before burning ...	131·15 c.c.	,, 21·4° C.	= 131·20 c.c.
After burning ...	130·19 c.c.	,, 21·5° C.	= 130·19 c.c.
After absorption of CO_2 formed ...	128·38 c.c.	,, 21·5° C.	= 128·38 c.c.
After absorption of oxygen ...	104·23 c.c.	,, 21·6° C.	= 104·20 c.c.

Pressure constant all the time.

CO_2 = 139·85 c.c. – 131·18 c.c. = 6·2 vol. %

Olefines = —

$$CO \ \& \ H = 131{\cdot}19 - 128{\cdot}38 - \frac{2{\cdot}80}{3} = 1{\cdot}344 \text{ vol. \%}$$

CO from iodine estimation = 1·228 vol. %

H therefore = ·116 vol. %

O = 128·38 – 104·20 + ·93 = 17·960 vol. %

Contributions and Correspondence.

SILVER ORE TREATMENT IN MEXICO.

(Additional note to contribution published in *Journal* of August, 1908.)

Some little time ago I received a request for a description of the preliminary treatment referred to in my contribution which appeared in the August, 1908, *Journal*, in discussion of Mr. Caldecott's paper on "Silver Ore Treatment in Mexico."

After making a large number of tests and applying every method that appeared to have any application whatever to the problem, I found only one means of unlocking the silver in the manganiferous ore so as to make it soluble in KCy solutions, and that was by preliminary chloridising. By treating with potassium chlorate and hydrochloric acid, or roasting with salt and sulphur, and following this with treatment by weak KCy solutions in the usual manner, a uniformly high rate of extraction was obtained. I desired however to hit upon a method that would be more feasible to develop on a practical working scale than either of these appeared to be, and it occurred to me to apply the common laboratory method of preparing chlorine, using salt, manganese dioxide and sulphuric acid according to the reaction :—

$$2NaCl + MnO_2 + 3H_2SO_4 = MnSO_4 + 2HNaSO_4 + 2H_2O + Cl_2.$$

The salt was mixed with the ore, crushed through 100 mesh screen. The sulphuric acid was added in weak solution (5% H_2SO_4) and allowed to stand with occasional agitation for 24 hours. The solution was then filtered off, the residues washed and treated with ·2% KCy solution. The silver extraction varied from 60 to 94%. The method has the advantage of low cost of materials; commercial sulphuric acid and salt are cheap, and the manganese dioxide is already present in the ore. I regret that I have not had the opportunity of going further and working out the practical application of the method used in these laboratory tests. The results were so satisfactory that I consider the method worthy of the attention of operators who have large enough bodies of refractory manganiferous silver ores to justify the expense connected with working out a new process.

Robert Linton.

Los Angeles, Cal.,
25th January, 1909.

FURTHER NOTES ON THE UTILISATION OF WASTE HEAT IN SLIMES TREATMENT.

By A. Salkinson (Member)

Since the publication of this paper in June, 1907, a number of mines have adopted the scheme with successful results; to those which are still considering the matter, the following notes may be of interest.

To begin with, the most suitable source of heat, though by no means the only one, is the mill engine: this has been very well utilised, amongst others, on the Knights Deep-Simmer East plant, where a solution heater has been erected between the exhaust of the mill engine and the condenser. The heater consists of a shell containing about 400 steel tubes, through which the solution is circulated from the sumps and so arranged that it has to pass four times through the heater before being returned. The plan adopted is to circulate the solutions alternately, so that only one heater is in use on each plant; every two months the tubes are cleaned, which is done without stopping the steam, the latter passing through the body of the heater to the condenser.

The principal advantage in utilising steam from the mill engine is that a large and constant source of heat is available; it also helps the engineering department, enabling the latter to obtain either a greater vacuum or to lessen the amount of water pumped through the condenser and cooling tower. Generally the principal objections met with are that the mill engine or other available source of heat is too far away from the slimes plant, and also that there is no waste heat. This latter astounding statement is not at all infrequent; however with the exception of one mine, which has electrical installation throughout, supplied with power from a distant source, it is of course groundless. Cyanide solution acts quite as well as a condenser of exhaust steam as water, and the results are far more satisfactory; but there is often a strong feeling against interfering with the existing plant.

As regards the distance from the available source of heat, this adds only a trifle to the cost of power, provided that the piping is not too small.

As will be seen from the figures below, the Simmer East has gained one-third in treatment capacity and one-fifth in extractor box capacity, as well as better precipitation. Nor must this be assumed as the limit, as the solutions could be still further heated, the end point being the temperature at which cyanide solution begins appreciably to decompose; solutions of a temperature of 130° to 140° F. show no signs of decomposition. A simple and cheap way of utilising exhaust steam from a non-condensing engine is to lead the steam directly into the solution sump. On the Wit Deep, a drum 4 ft. deep open at the top, and with an outlet just below the top has been attached to the side of the precipitated solution sump and the exhaust steam from a fan engine is led nearly to the bottom of the drum. The outlet pipes from the extractor boxes deliver into the drum and the solution overflows through the outlet, so that the steam has to pass through about 4 ft. of cold solution, which effectually condenses it. Hence the height of the solution in the sump is immaterial. Special attention may be drawn to the gain shown in the extractor boxes in the figures below, 20% more solution being precipitated per unit of zinc, and the percentage precipitation values having risen from 91 to 94·5%. Mr. Walford Dowling, reduction manager of the Knights Deep-Simmer East plant has informed me that since these figures were made up, the amount of solution precipitated has risen to a little over 2 tons per cub. ft. of zinc, or a gain of 30%.

It must not be forgotten that this would apply also to sands solutions, and a decrease of one-fifth to one third in extractor boxes and zinc is not to be despised. Another point of interest noticed on the Wit Deep is in connection with last drainings; after a sand charge has had its quota of weak solution, it is drained dry, and hot precipitated slimes solution is pumped on as a last wash. On comparing the last drainings, it was found that those from slimes solution were, contrary to expectations, almost invariably a little higher than from the cold weak solution. This can only be explained by the fact that hot solutions exert a quicker dissolving effect. It is hardly necessary to state that the original values of the two precipitated solutions were duly ascertained. On the Glencairn, heating of sands solutions has been carried out for some months and, I am informed, resulted in an appreciable lowering of the residues. The Village Main Reef successfully used the process on portion of their plant during last winter for a mixture of accumulated and current slimes and obtained a considerable saving in settling time.

I am indebted for the following figures to Mr. J. W. Craig, manager of the Knights Deep and Simmer East, Ltd., and they are published by his kind permission. The gain in the collecting tanks is not shown, as the temperature of the mill water had already been raised before the installation in the treatment tanks. The distance of the solution sumps from the heater is about 600 ft.

SIMMER & JACK EAST, LTD.—SLIME TREATMENT, WITHOUT AND WITH HEATED SOLUTION.

		Without August, 1908.	With Sept. 9, to Oct. 8, 1908.
No. of vats in use	Collecting ...	4	4
	1st settlement	5	3
	2nd ,,	4	3
Total tons of slimes treated ...		9,900	10,620
Tons of slimes per day ...		320	354
Average depth of charge in collecting vat		3′ 5·08″	3′ 8·61″
Average tons per charge at 47·5% moisture		165	179
Average temperature in Fah. degrees	Pulp during collection	70·3	73·7
	On completion of 1st transfer	58·9	86·4
	On completion of 2nd transfer	57·4	86·6
Average settling time, hours, 1st settlement		46·6	30·1
Average settling time, hours, 2nd settlement		44·6	28·2
Average time under treatment, hours		102·4	65·5
Average circuit in hours	Collecting ...	60	54
	1st settlement ...	75	41
	2nd ,, ...	60	41
Per cent. of charges	One settlement only ...	3·3	3·2
	Not completely settled		
	In 1st settlement ...	1·6	40·3
	In 2nd settlement ...	57·6	12·9
Estimated tons 2nd wash solution lost through incomplete settlement		787	158
Assay value of slime charges ...		1 486	1·501
Assay value of residues ...		0·237	0 245
Percentage extraction ...		84·0	83·7
Precipitation	Temperature entering extractors ...	57·3	77·3
	Tons solution per c.f.Zn. per 24 hours ...	1·555	1·89
	Assay value before ..	0·357	0·382
	Assay value after ...	0·032	0·021
	Percentage extraction ...	91·0	94·5

SIMMER & JACK EAST, LTD., SLIMES PLANT.

Notes on comparison of treatment without and with heated solutions :—

As no extra heat was added to the slime pulp at collection, the number of collecting vats remains the same in each case. The extraction in each case is practically the same; 720 tons more slime was treated with heated solution, in one day less, than was treated without heated solution, or 354 tons per day against 320.

Six vats did more work with heated solution than nine did without heat. This means 33·33% saving of plant without allowing for extra work done. Calculating the saving of plant from the settling time figures, the result is 36%, and on time under treatment figures also 36%. These latter figures are not quite reliable on account of the difficulty of keeping the solution low enough in decanting to see when the slime is settled. The more conservative figure had better be adopted. The incompletely settled charges for both cases were very near complete settlement, the difference averaging less than 3 in. per charge. The circuit hours calculation was based on a vat holding 200 tons of dry slime. The improved precipitation is very marked, more and better work being done with less zinc. Finally, it is safe to say that more heat would do quicker work.

Erratum.

ANALYSIS OF LEAD AND SILVER IN ORES.—Dr. J. Loevy writes pointing out a printer's error in the November *Journal* under the above heading.

Journal November, 1908, page 170, column 1, line 19, for "poor" read "pure."—Ed. Com.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

ANALYSIS OF ALUMINIUM POWDER.—Aluminium powder can readily be analysed by heating with an excess of ferric sulphate in an atmosphere of carbon dioxide, and determining the amount of ferrous salt formed by means of permanganate. The equation is $3Fe_2(SO_4)_3 + 2Al = Al_2(SO_4)_3 + 6FeSO_4$. In two analyses the author has found that nearly 6 % of oxide is present, but he cannot positively affirm that this oxide is aluminium oxide."—É. KOHN-ABREST, *Chemical News.—Forey Sciences*, January 15, 1909, p. 35. (E. H. C.)

THE CONTROL OF DISINFECTANTS.—In the *Lancet* of Sept. 19th, p. 902, Dr. S. Rideal and Mr. J. T. A. Walker draw attention to the necessity which exists for the examination of disinfectants after they have been allowed to stand after dilution for 24 hours. The subject is one of considerable importance to users of disinfectants, as frequently a disinfectant solution is made up and allowed to stand for several days before it is completely used, so if the dilution necessary is calculated upon a freshly made emulsion which afterwards separates and which in this way has a lower efficiency than that calculated, it is quite conceivable that considerable damage might be done. Some results which I recently obtained on a comparatively recent disinfectant with a guaranteed coefficient of 17 will be of interest. The sample was obtained from bulk which had been supplied to a large sanitary authority. The co-efficient on the freshly made emulsion was only 14, but when a 1 % emulsion had been allowed to stand for 24 hours the co-efficient had dropped to nine. This instance is only one of several which could be quoted, and I trust that those interested will adopt the suggestion of Dr. Rideal and Mr. Walker.—T. H. LLOYD, F.C.S.—*Lancet*, Oct. 17, 1908. (G. H. S.)

REPORT OF THE INTERNATIONAL COMMITTEE ON ATOMIC WEIGHTS, 1909.—The most important determinations which have been made during the year may be summarised thus :

Hydrogen. 1·00787. W. A. Noyes. By complete syntheses of water.

Chlorine. 35·457. Noyes and Weber. By syntheses of HCl : 35·468. Edgar. By syntheses of HCl.

Sulphur. 32·070. Baumé and Perrot. By determination of density of H_2S.

Lead. 207·19. Baxter and Wilson. From analyses of the chloride.

Tellurium. 127·609. Baker and Bennett. By heating TeO_2 with S ; 127·54. Baker & Bennett. By direct conversion of Te to $TeBr_4$; 126·65—126·94. Marckwald. By careful dehydration of telluric acid.

Rhodium. 102·9. Hüttinger. Reduction of pentamine chloride with hydrogen.

Palladium. 106·708. Woerule. Analysis of palladosamine chloride ; 106·75. Haas. Reduction of palladosamine bromide : 106·434 Kemmerer. Reduction of palladosamine chloride in hydrogen.

Europium. 152·03. Jantsch. Analysis of octohydrated sulphate.

Erbium (Neo). 167·43. Hofmann and Burger. Synthesis of sulphate obtained from oxide by repeated fractionations.

Ytterbium. 170·6—174·02. Urbain - Welsbach. The old element has been proved to be a mixture of two elements, called neoytterbium and lutecium, of approximate atomic weights 170 and 174 by Urbain and Welsbach working independently.

Columbium (Niobium). 93·5. Balke and Smith.

Radium. 226·4. Mme. Curie and Thorpe. Analysis of $RaCl_2$.

In their report for 1908 this committee recognised the fact that a general revision of the atomic weight table was desirable, and such a revision has now been made. Modern investigations have shown that the fundamental values required modification, and through them many other atomic weights are affected, although the changes thus brought about are less important than they were generally supposed to be. Many atomic weights remain practically unaltered, and in few instances are the changes large, as a comparison of the new table with its predecessors will show. A careful scrutiny of all the evidence was, however, none the less necessary, and the table now offered gives the results thus obtained.

The fundamental atomic weights, the standards of reference employed in the calculations, are as follows, when O = 16.

H	... 1·008		Ag	...	107·880
C	... 12·000		K	...	39·095
N	... 14·007		S	...	32·070
Cl	... 35·460				
Br	... 79·916				

The value for silver is possibly a trifle too low, by from three to five units in the third decimal place. A combination of the best measurements gives Ag = 107·883. In this case, and in others as well, the second place of decimals is given in the table, the third place being uncertain. Thus we have K 39·10, N 14·01, Br 79·92, etc. Only with hydrogen is the third place retained.

	Symbol.	Atomic Weight.		Symbol.	Atomic Weight.
Aluminium	Al	27·1	Molybdenum	Mo	96·0
Antimony	Sb	120·2	Neodymium	Nd	144·3
Argon	A	39·9	Neon	Ne	20·0
Arsenic	As	75·0	Nickel	Ni	58·68
Barium	Ba	137·37	Nitrogen	N	14·01
Bismuth	Bi	208·0	Osmium	Os	190·9
Boron	B	11·0	Oxygen	O	16·00
Bromine	Br	79·92	Palladium	Pd	106·7
Cadmium	Cd	112·40	Phosphorus	P	31·0
Caesium	Cs	132·81	Platinum	Pt	195·0
Calcium	Ca	40·09	Potassium	K	39·10
Carbon	C	12·00	Praseodymium	Pr	140·6
Cerium	Ce	140·25	Radium	Ra	226·4
Chlorine	Cl	35·46	Rhodium	Rh	102·9
Chromium	Cr	52·1	Rubidium	Rb	85·45
Cobalt	Co	58·97	Ruthenium	Ru	101·7
Columbium	Cb	93·5	Samarium	Sa	150·4
Copper	Cu	63·57	Scandium	Sc	44·1
Dysprosium	Dy	162·5	Selenium	Se	79·2
Erbium	Er	167·4	Silicon	Si	28·3
Europium	Eu	152·0	Silver	Ag	107·88
Fluorine	F	19·0	Sodium	Na	23·00
Gadolinium	Gd	157·3	Strontium	Sr	87·62
Gallium	Ga	69·9	Sulphur	S	32·07
Germanium	Ge	72·5	Tantalum	Ta	181·0
Glucinum	Gl.	9·1	Tellurium	Te	127·5
Gold	Au	197·2	Terbium	Tb	159·2
Helium	He	4·0	Thallium	Tl	204·0
Hydrogen	H	1·008	Thorium	Th	232·42
Indium	In	114·8	Thulium	Tm	168·5
Iodine	I	126·92	Tin	Sn	119·0
Iridium	Ir	193·1	Titanium	Ti	48·1
Iron	Fe	55·85	Tungsten	W	184·0
Krypton	Kr	81·8	Uranium	U	238·5
Lanthanum	La	139·0	Vanadium	V	51·2
Lead	Pb	207·10	Xenon	Xe	128·0
Lithium	Li	7·00	Ytterbium		
Lutecium	Lu	174·0	(Neoytterbium)	Yb	172·0
Magnesium	Mg	24·32	Yttrium	Y	89·0
Manganese	Mn	54·93	Zinc	Zn	65·7
Mercury	Hg	200·0	Zirconium	Zr	90·6

In adjusting the other atomic weights the determinations by Richards and his colleagues have generally been given preference. They are certainly entitled to the highest weight, but probably not to exclusive consideration. The work of Guye and his associates at Geneva, and the recent direct measurements of the chlorine-hydrogen ratio are also of very great importance. It is to work of this order that we must look for ultimate precision. Important investigations upon atomic weights are now being carried on in several laboratories, and our knowledge of these constants will doubtless become much more exact within the near future."

F. W. Clarke, W. Ostwald, T. E. Thorpe, G. Urbain.—*Journal of the American Chemical Society*, Jan., 1909, xxxi, p. 1. (J. A. W.)

Test for Purity of Mine Air.—"Dr. Angus Smith, in a report to Lord Kinnaird's commission, gives it as his opinion that the quantity of CO_2 in the air of mines should never exceed ¼, or ·25%.

Dr. Smith gives a very simple test by which determination can readily be made to determine whether mine air contains as much, or more, or less quantity of CO_2 than ¼%.

The necessary apparatus consists of: One 8-oz. bottle and cork; one 5-oz. bottle and cork; one bottle of limewater with excess of lime; one pipette or measure holding ½ oz.; four ½-oz. bottles corked; one ¼-oz. bottle, containing an alcoholic solution of phenolphthalein; one piece of india-rubber about 1 ft. long.

The method of preparing for and making the test is as follows: Measure ½ oz. of clear limewater and pour into a 5-oz. bottle, and fill up with distilled water, giving a solution of 10% strength. Add a drop or two of the phenolphthalein solution; the limewater assumes a pink colour. Now fill the 8-oz. bottle with mine air from a sample flask which has been obtained from the mine, add ½ oz. of the coloured solution and shake. If the pink colour disappears quickly, the air contains more than ¼% of CO_2. If the colour fades only after repeated re-shaking, the quantity of CO_2 in the air is approximately ¼%."—*The Chemical Engineer*, Dec. 1908, p. 269. (W. A. C.)

Methods of Testing the Efficiency of Ventilation.—"A general consensus of opinion among recent investigators points to recorded temperature as the most practicable means of controlling ventilation under modern conditions. The temperature of any given room may increase by any one or any combination of the following causes: (1) Heat given off by the bodies of the occupants and by their breath; (2) Heat given off by the lights; (3) Rise of outside temperature; and (4) An excess of heat from the heating plant.

The last of these causes should be practically eliminated in a modern, well-constructed, and well-managed public structure. In all such places the heating system is, or should be, designed to deliver a constant amount of heat for any given period of time.

Contrary to the old theory that a heating plant should "keep a room at a given temperature," and so automatically supply less and less heat as the heat derived from the occupants increases, the new idea is rather that the plant deliver a constant predetermined amount and that the ventilating plant keep the room at the desired temperature. This idea is really carried out in most of the buildings with which the ventilating engineer or inspector would have to deal. This being the case, any rise of temperature in a room would be caused by the animal heat of the occupants, effect of the lights, or by rise of the outside temperature. This last cause is slight in any instance and may usually be neglected.

Then, since in giving off heat the body and the lights (with the exception of electric incandescent) give off both CO_2 and moisture in proportion, we have in this increase in temperature an almost ideal index of the quality of the air. A recording thermometer with a large dial face, hung or placed so as to be as readily seen as a clock, is the best known gauge of an occupied room.

If the air current through an occupied space is sufficient to carry away the increased moisture and to keep the temperature even, it will lower the carbon dioxide in the same proportion This may be accepted as true without chemical examination (there are occasionally other sources of CO_2). The humidity is therefore an excellent indicator of the degree of ventilation, and a recording hygrometer is most desirable to place alongside the recording thermometer.

We feel safe in stating that probably the best engineering practice in ventilation to-day is in the use of these instruments, combined with training in the detection of odors. Only in cases of control or inspection will the determination of CO_2 be necessary when these instruments are at hand and when time for observing them is available. But there are frequent circumstances in which CO_2 is an equally good indicator and the tests for it more convenient of application, but they should partake of the rapidity and comparative accuracy of the instruments, and should not require the engineer to be burdened with cumbersome apparatus. For these purposes a sufficient degree of accuracy may be defined as within two-tenths to three-tenths of a part per 10,000, with a possible maximum error of five-tenths

Interpretation of Results.—If in a crowded hall the temperature is less than 70° F., if the wet and dry bulb thermometers are 7° apart (less than 70 per cent. humidity), with no appreciable odour, and if the carbon dioxide is less than seven parts per 10,000, the air may be said to be satisfactory; but if the temperature is 80°, the wet and dry bulb only 4° apart (humidity 83%), and the carbon dioxide eight parts, with a close odour, then measures should be taken at once to remedy the condition.

The ideal condition for constant occupancy is: thermometer 68°, humidity 68%, or wet and dry bulb 6° to 7° apart, with carbon dioxide less than six parts per 10,000 of air.

If these general results may be obtained, the smaller differences in readings may be discarded until the public is educated to this extent."—Richards, Wade, Gilbert, Hanson and Talbot.—*Technology Quarterly*, Sept., 1908, p. 321-331. (E. H. C.)

Hard Chemical Products.—"The diamond, so long the hardest known substance, now has two rivals, the silicide and the boride of titanium—products of Prof. Henri Moissan's electric furnace—each being claimed to be as hard as the diamond."—*Mining World*, London.—*The New Zealand Mines Record*, Dec. 16, 1908, p. 221. (A. R.)

Zinc in Zinc Dust.—"In determining the amount of Zn in zinc dust by the volume of H evolved on treating with acid, it has been found that the use of rubber connections in the apparatus must be avoided, otherwise the results are too low. The phenomenon seems to be due to diffusion."—De Koninck, *Bull. Soc. Chim. Belge*, xxii., 113.—*School of Mines Quarterly*, Nov. 1908, p. 52. (H. A. W.)

Colorimetric Determination of Dissolved Oxygen in Water.—This method depends on the change in the colour of cuprous ammonium chloride when brought into contact with oxygen. The salt is difficult to prepare quite colourless, and hence it was advisable to construct an apparatus in which a solution of the pure colourless salt could be prepared and kept at all times ready for use. This was finally done in an apparatus where the rubber connectors were protected from air by an atmosphere of hydrogen, but even then the reagent could not be kept colourless for any length of time and finally an apparatus made entirely of glass was used. As soon as the water containing oxygen was brought into contact with the reagent the blue colour appeared and the comparison was made with a standard solution of cupric chloride of such strength that 1 c.c. of the solution was equivalent to 1 c.c. of oxygen in a liter of water when the quantity of of water taken for analysis was 100 c.c. This was prepared by dissolving 1·1364 grams of pure copper in aqua regia, evaporating off the excess of acid, dissolving in water and making up to 1 liter. 1 c.c. of this solution is equivalent to 0·1 c.c. of oxygen as indicated thus

$$Cu_2\,Cl_2 + 2HCl + O = 2\ Cu\ Cl_2 + H_2O.$$

The apparatus requires some skill to use it but when the necessary experience has been obtained accurate determinations of oxygen may be easily and rapidly made.—G. B. FRANKFORTER, G. W. WALKER and A. D. WILHOIT.—*Journal of the American Chemical Society*, January, 1909, xxxi. 35. (J. A. W.)

COBALT TIN ALLOYS.—A thermal and micrographic examination of alloys of cobalt and tin showed that these metals form two compounds, CoSn and Co_2Sn, both of which are non-magnetic and are considerably harder than pure cobalt. The compound, Co_2Sn, melts at 1171° C, but the compound, CoSn, decomposes at 927° C. before it melts. —S. SHEMTSCHUSHNY and S. BIELYNSKI, *Z. anorg. Chem.*, 1908.*59*, 364—370. *Chem. Zentr.*, 1908, 2, 1498.—*Journal of the Society of Chemical Industry*, Dec. 15, 1908.

METALLURGY.

CLASSIFICATION FOR TUBE-MILLING.—"The common practice on these fields for separation from the tailings pulp of coarse sand for regrinding has not varied from the form of hydraulic classification supported by the writer some years ago, in a paper contributed to the proceedings of another technical Society.* Other methods, which have been advocated, consist in separating the coarser grains for regrinding by means of a screen, as, for instance, of the Callow endless travelling belt type, or by Wilfley tables as on two mines on the Eastern Rand, or by continuously removing the heavier product settling in a shallow box by means of an inclined scraper conveyor.† This last method has the advantage of not requiring fall, and appears suitable for small installations; it is used in the United States in the form of the Dorr classifier,‡ which is fitted with reciprocating rakes.

As is well known, classification prior to tube-milling is here commonly effected by leading the underflow from several pyramidal or conical spitzlutten, by means of a launder, into a dewatering spitzkasten placed above and near the tube-mill inlet, which receives the finally thickened pulp underflow. This system is fairly satisfactory, but has the objection of requiring considerable elevation of pulp and a certain consumption of mill service water to assist clearance of the spitzlutte, whilst there are for each tube-mill a number of classifiers requiring adjustment, and each liable to clogging of the comparatively small nozzle outlet; further, constant attention is required to ensure a regular feed to the tube-mill of a pulp containing the desired percentage of moisture.

To overcome these difficulties there has lately been developed on the Simmer and Jack plant a method which is now in regular operation, and which offers considerable advantages. Briefly, it consists in combining the spitzlutten and dewaterer in one classifier, which receives the pulp direct from the tailings launder, and delivers its coarse sand underflow into the tube-mill. This classifier is of the conical type, with peripheral launder for receiving the final pulp overflowing to the cyanide plant. In its operation, a sufficient quantity of pulp, including return from tube-mills, after elevation is delivered from the main tailings launder by means of a by-pass and vertical inflow pipe, fitted with a horizontal baffle, slightly below the surface of the pulp in the centre of the classifier. Whilst smaller dimensions can be employed, the classifier at present in use is 6 ft. diameter at top and 9 ft. deep. This classifier is kept nearly filled to the top with sand, and an essential patented feature in its satisfactory operation consists of an internal serrated or notched horizontal disc diaphragm near the bottom. This ensures a steady underflow of thick pulp, containing very little moisture, which can be regulated in amount up to 440 tons of solids per 24 hours. An illustration given shows a classifier, with one of the various forms of diaphragm. Owing to the thick consistency and slow velocity of the pulp, a large underflow opening (2⅜ inches in diameter) is used, which wears but little and can be exactly regulated by an adjustable horizontal cut-off gate. To thin down the pulp for satisfactory tube-milling, turbid water is withdrawn from the upper portion of the tailings stream and mixed with the underflow entering the tube-mill, whilst a further supply from the same source is employed to further thin down the tube-mill outflow prior to amalgamation on shaking tables. By regulating the size of underflow opening, the tonnage of solids issuing can be altered without changing its percentage of moisture, which last is regulated for tube-milling purposes by the independent supply of turbid water. Should the volume of tailings pulp vary, the classifier automatically adjusts itself to the altered conditions, by the increased or decreased volume of pulp flowing upward to the surface of the classifier, sorting out and carrying away in the overflow larger or smaller grains corresponding to its changed velocity. The diaphragm, by supporting a considerable depth of settled sand, prevents any change in consistency or breaking away of the thick, sluggish underflow, whilst the large size of the opening precludes choking In practice on the Simmer and Jack about 440 tons of solids containing 65% + 60 (0·01 in.) grade are delivered per 24 hours from the classifier into the tube-mill as an underflow pulp containing 26% to 28% of moisture. The classifier overflow of final pulp at present contains about 10% + 60 (0·01 in.) grade, but of course varies with tonnage of ore milled.

The advantages of the system described are:—(*a*) Much less elevation of pulp for the tube-mill circuit so that the installation of an elevating system to this height only is contemplated, with subsequent elevation of the final pulp alone to the necessary height for delivery to the cyanide plant; (*b*) considerable economy in initial cost of construction, owing to several classifiers with fittings and accessory launders being replaced by a single classifier for each tube-mill; (*c*) simplicity of operation of the single classifier in place of several; (*d*) the dispensing with a supply of mill service water and its pumping cost; and (*e*) non-liability to derangement by choking, owing to large size of underflow outlet.

---

* See *Trans. Inst. Min. and Met.*, p. 55, vol. xiv., 1904.

† See MacDermott and Duffield's "Amalgamation of Gold Ores," p. 32, 1908.

‡ See *Mines and Minerals* June 1908, p. 541.

The use of deep conical hydraulic classifiers with diaphragms had been originally developed in connection with pulp classification for another purpose, but their successful application to tube-milling practice is largely due to observant attention and skilful modification by Mr. G. O. Smart, Reduction Works Manager of the Simmer and Jack Proprietary Mines, Ltd."—W. A. CALDECOTT.—*Journal of South African Association of Engineers*, Dec., 1908, p. 101. (W. R. D.)

THE CALLOW SCREEN.—"In most concentration mills the sizing or classification of ore in some manner is essential. The general custom has been to use either revolving or shaking screens for the coarser sizes, and water classification for the finer particles. With the ordinary revolving screen, or trommel, the limit of practical economical work is about 2·5 mm., or, say, one-tenth of an inch screen opening; finer than this, water classifiers are used. But water classifiers are generally admitted as a very poor substitute for screens, since they are quite different in effect, giving grades of equal falling particles, which is an entirely different thing to grades of equal-sized particles, and as a consequence water classification is not a proper preparation for concentration, while it is, on the other hand, most objectionable as introducing great excess of water into the slimes. Many and varied attempts have been made to invent a practical method of fine screening wet, and the failure so far resulting has been due to excessive wear, imperfect screening, clogging of mesh, or small capacity, with undue water consumption. The Callow travelling belt screen, it is claimed, is free from the objections named, and is one of those simple inventions which appeal to the practical man at first sight. The repeated attempts to make a practical fine wet screen are clear confirmatory evidence of the recognised large field of utility.

Water classification tends usually to make the fine slimes poorer than a similar grading (if possible) by screens would do, but it never makes the slimes poor enough for throwing to waste, and it increases the difficulty of their treatment by the introduction of a great volume of extra water. On the other hand, it is possible, on certain ores, to use Callow screens so fine that what passes through them is actually an enriched product, owing to the percentage of brittle mineral broken down fine in the crushing. For example: a copper slime which from water classifiers would assay 3% of metal, may by its production from a screen of 150 mesh assay 7½%, and thus become a smelting product added to the other mill concentrates, and therefore not subject to the great loss which all millmen know will follow the concentration of the 3% slime.

In the grading of material for recrushing—as for instance for tube mills following stamps—screening in certain cases offers great advantages over water classification; because in water the particles of heavy mineral which settle are much smaller than the particles of gangue rock, and thus much of the mineral already fine enough is subjected to regrinding, and it may be unnecessary, or even objectionable, in subsequent treatment, that the heavy mineral should be reduced much finer than the gangue accompanying.

The fundamental principle of this machine is a travelling band or belt of screen cloth, over which the ore and its carrying water is spread by means of distributing aprons. The band or belt of screen cloth passes over head and tail rollers, and the belt is caused to travel continuously in a horizontal direction at a speed varying from 25 to 125 ft. per minute, according to the nature and quality of the material to be screened. These aprons play a most important part in the operation of the machine, and its successful and proper operation depends in a great measure upon this feature and its proper construction and adjustment. It is with this that a preliminary sorting or sizing is done, for as the particles which are of all sorts and sizes fall from the lip, the coarsest and largest pieces, having the greatest trajectory, strike the screen at a point far ahead of the smaller ones, and they are there deposited, leaving a space behind free and unincumbered. The pores of the cloth in the rear being, therefore, still uncovered and open, permit the free passage of the fines and water. Simultaneously with this action the cloth is being moved forward and the deposit of oversize is continuously and unintermittently removed from the separating or screening zone, and thus the machine continues to perform its function so long as it is kept moving and is supplied with feed.

After leaving the screening zone, the deposit of oversize is carried forward and passes under a shaking spray of clear water, where any remaining traces of slime or of fine adhering particles are washed, and pass through with the water from the spray into the undersize hopper beneath. Continuing on, the oversize still clinging to the screen cloth, passes in front of a small impinging spray, which is conveniently situated somewhere about the mid-diameter of the front roller, and the oversize is there washed off the screen cloth into the oversize hopper below. The 24 in. Duplex machine is in two parts, each half being the duplicate of the other; the two belts are independent of one another, but operate from a common driving mechanism. The speed of the two belts can be varied by means of cone pulleys, and the driving shafts are arranged so that either side can be thrown in or out by suitable friction clutches adjusted to each driving roller. This permits of one side of the machine being shut down for changing or the repair of the screen cloth, all the feed being turned on the running side pro tem., it only being necessary to speed up the belt sufficiently to carry the increased load.

The rollers revolve on overhung shafts, so there are no out-board bearings to prevent the quick renewal of worn-out cloths, the new ones being slipped on (after being made endless) over the ends of the rollers. The screen belts are inserted on each side, between the plies of a 2-in. two-ply rubber belt; these rubber edges serve as a belt to drive the screens, and the flanged edge serves to confine the material to be screened to the screen belts, and prevents unscreened material escaping over into the undersize hoppers.

The shafts which carry the rollers are carried in long hubs or bosses attached to each side of a central casting or girder; to this casting are also attached the undersize hoppers. This casting is hollow, and forms, besides a support for the moving and all other parts of the machine, a central gutter or launder into which the combined undersize from each half of the machine discharges."—*Transvaal Leader*, Feb. 2, 1909. (A. R.)

NON-EXPANSIVE ALLOY.—"A new alloy which has been brought out in France, and is know as 'Invar,' has the valuable property of having practically no expansion, and is thus adapted for various purposes. M. Guillaume succeeded in forming such an alloy in connection with the Commentry-Decazeville metallurgical firm. Nickel steel is the base of the alloy, and up to 20% of nickel, the expansion is about the ordinary. It then increases up to a value of 24%, reaching a minimum value at 36% of nickel, then rising and taking the normal value for a 50% alloy. It is the 36% metal which has the curious properties brought out by M. Guillaume, and its expansion is almost zero, being 17 times less than that of steel and about the same as for melted quartz. The new steel is not magnetic, and it does not rust. Owing to these properties, it is already used at Paris and elsewhere in the construction of spiral springs, pendulums, graduated scales for instruments. Its field of usefulness is extensive, and it can receive many applications where these properties are needed. One valuable application is in the formation of standard measures, such as the metre, and different standard scales have been made at Paris by the International Bureau of Weights and Measures. Such scales or gauges are remarkably exact, owing to the non-expansion of the metal. They are used in the artillery works at present, and very precise work can be carried out with the gauges and calipers, diminishing the errors which were found previous to their use. According to the tests made upon standard scales, these hold their value within a remarkably small percentage. By varying the amount of nickel in the combination, a second alloy can be formed which is valuable as having the same coefficient of expansion as glass, and it can thus be used to replace platinum in the leading-in wires of an incandescent lamp, thus reducing the cost of these wires to a comparatively low figure. It can also be used to replace the platinum wires of eudiometers and Geissler tubes for the same reason."—*Scientific American Supplement.*—*Mining Science*, Dec. 31, 1908, p. 523. (A. R.)

DANGERS OF WHITE ARSENIC.—"At a meeting of the Cape Chemical Society, Professor B. de St. J. van der Riet, of the Victoria College, moved that in the opinion of the Society the present law regulating the sale of poisons should be so amended as to restrict the sale of uncoloured arsenical preparations to the public. There was, he said, a law regulating the sale of poisons, but it did not seem to prevent white arsenic or arsenical preparations coming into the hands of ignorant and careless persons. At present, also, criminals obtained pounds of white arsenic with the greatest ease, and people were in the habit of leaving the substance lying about houses in the most careless manner imaginable. He thought that regrettable accidents and crime could be prevented by the Colony adopting the English law on the subject, which compelled the colouring of all arsenical preparations sold to the public, so as to take away their unsuspicious appearance.

Dr. C. F. Juritz stated the English law compelled the addition to arsenic and arsenical preparations of at least six per cent. of soot or three per cent. of indigo.

The motion was unanimously agreed to, and it was further decided that the Council of the Society should communicate the resolution to the Government."—*Rand Daily Mail.* (G. H. S.)

TOOL STEEL.—"It was, of course, well known that to subject ordinary carbon tool steel to such temperatures as 2,000° F., would mean its absolute ruination, and it was, therefore, evident that carbon, the most important constituent in ordinary tool steel, was only of secondary consideration in self-hardening steel, and that it was by virtue of some other special constituent that the latter could retain its hardness at high temperatures.

Further experiments on these lines soon made it clear that it was the proportion of tungsten, and in a lesser degree, that of chromium, that makes the difference between the ordinary self-hardening steel and high-speed steel. By working on this knowledge, and making many patient and costly experiments, steel manufacturers have now placed on the market a high-speed tool steel which gives results that are nothing short of marvellous, when compared with the results obtained from ordinary carbon and self-hardening tool steels before the introduction of their formidable rival.

The following analyses show clearly the difference in constitution of the three different classes of tool steel that have been referred to :—

	Ordinary Carbon Tool Steel.	Self-Hardening Tool Steel.	High-Speed Steel.
	%	%	%
Carbon ...	1·04	2·4	·67
Tungsten ...	–	5·62	18 90
Chromium ...	—	·49	5·47
Manganese ...	·25	1·90	·11
Silicon ...	·145	·71	·034
Sulphur ...	·012	·031	·029
Phosphorus ...	·022	·035	·013

A great drawback to the high-speed steel of the early days was the cumbersome method of hardening the tools made of it. To obtain the best results it was necessary to heat the tool to a white heat and quickly reduce it to a dull red, which was accomplished by plunging the nose of the white-hot tool into a vessel containing molten lead. This, coupled with the difficulty of persuading tool smiths of the necessity of bringing the tool to a white heat in the first instance, led to the system of returning tools to the makers when re-forging and re-hardening became necessary. Such difficulties are, happily, no longer a bar to development of the use of high-speed steel. Improvement in manufacture has rendered the handling of a high-speed tool a comparatively simple process, and time has overcome the very natural reluctance of the tool smith to abandon that experience which, prior to the advent of high-speed steel, had taught him that tool steel heated beyond a bright red was absolutely ruined."—R. E. L. MAUNSELL.—*Page's Weekly*, Dec. 18, 1908, p. 1243. (A. R.)

THE ABSORPTION OF WATER BY COKE.—"A series of researches on the power of red-hot coke to absorb water are detailed in the issue of *Stahl und Eisen* of January 6, and the results are set forth in graphic diagrams. The amount of the water again given off at various intervals of time is also shown. It would appear from the results of these tests, which were conducted with small quantities of coke (1 kg.), that red-hot coke plunged into cold water will absorb from three to four times the volume of water as compared with the quantity which it will take up, if it is placed in hot water when it is cold. The results of ten experiments are also recorded, in which freshly drawn red-hot coke, in quantities of about 100 kg. at a time, was immersed in cold water and allowed to remain therein for half an hour. It was then withdrawn and placed in casks with a perforated bottom, out of which the water was allowed to drip for an hour. Subsequently the weight of the coke was ascertained at intervals of 24, 48 and 72 hours—that is to say, the gradual evolution of water was ascertained. Ultimately the whole of the coke was stamped to powder and carefully dried, so as to ascertain the weight of the coke alone. The test proves that slightly damped coke will give off water in a railway wagon at the same rate as that which has been more drenched. Samples, both of foundry coke and of blast furnace coke, were tested."—*Times Engineering Supplement*, Jan. 13, 1909. (J. A. W.)

Sparks from Alloys.—"Auer von Welsbach has recently discovered that certain alloys of iron with cerium, lanthanum and other rare metals obtained from monazite, emit brilliant sparks when scratched with a knife or file. When strongly pressed with a piece of steel the alloy gives forth a brilliant flame that is accompanied by little heat or smoke. The sparks readily ignite gases, tinder, alcohol wicks, and explosives, such as are used in blasting and gunnery. The flame might also be made serviceable in flashlight photography, dispensing with the disagreeable odor of magnesium powders, and another possible use suggested is for military signalling."—*Mining Science*, Dec. 10, 1908, p. 463. (A. R.)

MINING.

Mount Morgan Inquiry Report.—"The report of the special board appointed to inquire into the disasters in the Mount Morgan mine on September 5 and in November was tabled in the Legislative Assembly by the Minister for Mines recently. The board makes the following recommendations:—

(1) The abandonment in the copper chambers of the pigstye system of timbering, as carried out in the past.

(2) The adoption of a sectional system of mining in all such parts of the mines as are already known, or may reasonably be suspected to be intersected by dykes or 'heads.' It was added that the system proposed by the mine manager (Mr. James White) appeared to be such as would meet the requirements in parts of the mine adapted to its use, it being a system that would lend itself more generally to the mining of ore bodies, such as are at present exposed in the copper chambers; and inasmuch as by this method a greater extent of ore could be worked at one time.

(3) Whilst recognising that no one system could be recommended for working the whole of the ore bodies, the board advised the careful selection of suitable methods of working different parts, with a view to securing a maximum of safety, a matter which, owing to the sudden variations from a hard compact ore to that of a treacherous nature, must of necessity be left to the judgment of the mining engineer.

(4) The exercise of the greatest precaution in the inauguration of any method of mining adopted in the future, as they recognised that the process of changing from one system to another would take some time, and would be attended with considerable risk.

(5) That in all work done in treacherous ground in the future reliance should be placed on close filling rather than on timber as a support. The filling available was of very good quality, and thoroughly suitable for this purpose.

(6) That more attention should be paid to the mullocking and filling of depleted stopes, and that a rigid course of filling those at present open should be initiated from and including the sill floor; this system of filling should be carried on concurrently with the work and kept up to within a workable distance of the face.

(7) That using wedges for blocking timber to the back, walls, or face, more attention should be paid to keeping the wedges tight; in the past there appeared to have been some misconception as to the use of wedges for taking up shrinkage and carrying weight, as distinguished from their use as a means of showing pressure.

(8) The Mining Act of 1898 did not make compulsory the reports of falls of ground unless these were attended with serious injury to any person. The reporting of all falls and movements of ground was necessary, and could be provided for by amending legislation upon the lines laid down in the evidence of Mr. Inspector Ward, a course which the board recommended.

(9) The board was satisfied that the Inspector of Mines had been attentive to his duties, and carried them out satisfactorily. They did not consider necessary the appointment of a resident inspector of mines at Mount Morgan.

(10) The replies to certain questions pointed to the fact that the duties previously carried out by three officers have of late devolved upon the registered mine manager, and they recommended that the attention of the company be called to the matter.

(11) That the workings of any flat-backed stope with two faces advancing towards each other be discontinued, unless the intervening ground is mullocked close up to the back.

(12) That in adjoining sections of the mine, where square set work is adopted, the timber should be laid out in its relatively correct position, so that it may ultimately be connected, and may thus develop its full strength.

(13) The board recommended a continuance of the use of ironbark, spotted gum, or other approved hardwood timber for square sets. As regards brigalow timber, in spite of the objections of a number of witnesses the board took no exception to its use except for the purposes for which it has been used in the mine. In conclusion, the board acknowledged the assistance rendered to them throughout the course of the inquiry by the management, shift bosses, and miners."—*Australian Mining Standard*, Dec. 30, 1908, p. 747. (A. R.)

Amount of Coal Dust in Airways.—"In a paper read before the Institution of Mining Engineers, in Edinburgh, Henry M. Hall, H.M. Inspector of Mines, gave an account of the various investigations in regard to the explosive properties of coal dust and the findings of the Commission on Explosives on Coal Dust in Mines. Continuing, he said:

Before any useful suggestions could be made as to the treatment of coal dust, so as to render it innocuous, it was absolutely necessary to obtain some idea as to the rate at which dust is deposited on underground roadways. Having gained this infomation, at any rate, approximately, it then becomes possible to judge fairly accurately of the practicability of any suggested remedy.

The writer believes that hitherto nothing has been done to obtain this information. Dusty roads, even to the extent of coverings several inches thick, are familiar to all mining engineers, but over what period of time these coverings have been accumulating no evidence is forthcoming; in some cases it may have taken only days, in others months, or even years. Aided by the willing help of others, the writer has set himself the task of solving this question, at any rate so far as to obtain a working idea of the rate of the accumulations in distinct parts of the mine, and under ordinary mining conditions. With this object, aspirators were fixed in a return air road at intervals of about 75 ft.

The action of the aspirator is by means of cotton-wool; the air is drawn through it and the dust deposited in transit, the cotton wool being carefully weighed before and after the operation. A bottle on a stool is filled with a measured quantity of water, which is then siphoned out into a bottle on the floor, thus allowing the air to pass through a tube which contains the cotton-wool, and deposit the dust.

As a result of these experiments, it was shown that something like 11½ lb. of coal dust were carried in eight hours past the point where the measurement was taken. The velocity of the current of air carrying this dust was 95 cub. ft. per minute. The measurements for dust in the air were made with a bellows aspirator, and it was found that in 3 cub. ft. of air the amount was ·002, ·001, ·002, and ·0005 gm., depending on the point of measurement.

These results show that there was gradual diminution of the dust as it travelled outbye.

A third method of measurement was adopted, with the object of arriving at the quantity of dust deposits on the floor, sides, etc., as it was being carried along by the ventilating current. Flat porcelain dishes of an area of 1·12 sq. ft. were used, two of which were placed at points on the main intake and two on the main return. These were in operation for periods varying from 8 to 10 hours on two separate days. The dust passing the two points on the main intake must have come partly from the screens and partly from the tubs passing up and down the shaft (about 600 tons of coal passed up the shaft each day).

The results show that at the shaft bottom, where the area of the road is 150 sq. ft., something like 11·47 gm. of dust were passing in the air each minute when the velocity was 225 ft., that is, 18 lb. in 6 hours. So that at least this quantity is being deposited each working day on to the floor and ledges of the main road between the bottom of the shaft and the bottom of the tunnels, 3,000 ft. inbye. If it be assumed that this road averages 12 ft. in width and that ½ lb. of dust per linear foot of such a road is required to make an explosive mixture of dust and air, starting with a clean sheet, it would require 83 ordinary working days to render this 3,000 ft. of road absolutely dangerous. (Note.—The deposit, of course, will be thicker near the shaft than farther in.)

An interesting experiment was made at Garswood Hall colliery. The screens at this colliery are near the downcast shaft, and a considerable quantity of dust is made in the process of screening, some of which finds its way back into the mine along with the ventilation. To obviate this, water sprays have been fitted up at the bottom of the shaft, and a frame has also been put in the main airway a little distance from the shaft; arranged around this frame are several sprays, the object of which is to wash the dust from the air as it passes through the frame. Quite a considerable amount of dust is thus washed out. With 40,000 cub. ft. of air passing through, 7½ lb. (weight after drying) of dry dust was recovered by means of a settling tank in 14 hours, and probably as much more escaped collection, which means that the sprays washed down 14 or 15 lb. in the 14 hours. This dust would originate partly at the screens and partly be blown from the trams in the pit shaft. The total quantity of dust finding its way back on the road-ways must be very large, when with only one frame so much can be recovered.

These experiments, made with a view of getting an idea of the rate of accumulation of dust in mines under ordinary working conditions, should be carried much further; but, so far as they go, it seems clear that the common estimate of the quantity of dust deposited day by day far exceeds the actual facts; and it follows that when once the roads have been made thoroughly clean, to keep them so ought not to present any insuperable difficulty. Allowing dust to get back into the workings from the shafts and screens amounts to poisoning the ventilation before it starts on its duty."—HENRY M. HALL.—*Mines and Minerals*, Dec., 1908, p. 223. (A. R.)

EUCALYPTUS FOR MINE TIMBERS.—"The woods of the eucalyptus are heavy, only a few species floating in water. According to McClatchie, there are 22 species cultivated in the United States that are suitable for railroad ties and underground work. Selection of species with reference to use as timber must be considered in relation to soil and climatic requirements.

To the average person, the term 'eucalyptus' has meant nothing but the most widely planted species, *Eucalyptus globulus*, or 'blue gum.' In Australia several species are known as 'blue gums' and several as 'red gums'; and the same species may be designated as by several different common names. Australian common names that have been generally adopted in America are 'blue gum tree' for *Eucalyptus globulus*, and 'red gum tree' for *Eucalyptus rostrata* and several other species indiscriminately. The entire genus *Eucalyptus* should not be condemned because the wood of one species, according to Mr. T. L. Carter, has been found inferior to other timbers for mining purposes. The principal drawback to eucalyptus timber has been its tendency to check, or open fissures and shakes, and its hardness, with consequent difficulty in spiking and working. The better understanding of methods for seasoning and handling this timber is doing much for its advancement. Mr. Carter has considered only *Eucalyptus globulus*, and has hardly given that widely known species its due. Its timber is hard, heavy, strong, and fairly durable. In America, however, it is not considered equal in timber properties to the ironbarks, to the sugar gum, or to the red gums. Its fast-growing characteristic has been its chief claim for prominence, and has led to planting in different parts of the Southwest for windbreaks and fuel,

For mine-timbering there are other species that are superior to the blue gum. There are about 17 species of eucalyptus that could be grown by mining companies and would furnish satisfactory material for timbering. According to the report of the State Board of Forestry, blue and sugar gum timber have been used with success in timbering mines near Escondido that are flooded during a part of each year.

The sugar gum (*Eucalyptus corynocalyx*), the red gum (*Eucalyptus rostrata*), and the narrow-leaved ironbark (*Eucalyptus crebra*) are eucalypts that should recommend themselves to the mining industry.

The durability, hardness, and strength of the timber obtained from these species make them eminently fit for the mines. The sugar gum furnishes a wood that is very durable as railway ties, as posts, and for underground use. Baron von Mueller states that posts set in the ground 15 years showed no signs of decay. The wood warps but little in drying, and when dry is very hard. In the interior valleys of southern California and Arizona it withstands both the intense heat of summer and the frost of winter. For desert country this is one of the most eligible among timber eucalypts. It does not grow quite as rapidly as the blue gum, but furnishes a more valuable timber. In New South Wales the ironbark is considered the king of hardwoods. The narrow-leaved ironbark endures a greater variety of climatic conditions than do other ironbarks. At Fresno, California, it grows vigorously, and young trees have grown well near Phœnix, Arizona. It is said to be content with poor soil, and, judging by experience with it thus far, it ought to grow in most valley and hillside situations in the Southwest.

An important point in considering the value of commercial plantations of eucalyptus has been determined by the United States Forest Service, from strength tests made by them, namely, that the fastest growing species are also the strongest. The bending tests showed the modulus of rupture in pounds per square inch to be:

	Age in Years.	Pounds.
Sugar gum	15	25,334
Blue gum	30	23,265
Blue gum	15	16,900
Red gum	15	14,380

The crushing strength in pounds per square inch, the compression being parallel to the grain, was:

	Age in Years.	Pounds.
Sugar gum	15	11,290
Blue gum	30	12,310
Blue gum	15	8,190
Red gum	15	7,723

The crushing strength of some American woods, in pounds per square inch (from Kidder):

	Pounds.
Oregon pine	4,500
California redwood	3,000
White pine	3,500
White oak	4,000

Comparison with Forest Service tests on hickory shows that 30-year blue gum is stronger than XXX hickory, and that 15-year sugar gum is nearly as strong as second-growth hickory. These tests would seem to disprove Mr. Carter's statement that the blue gum wood is of poor quality, and not strong enough for props in heavy ground. Perhaps the timber was not properly seasoned, or from immature trees. We see from the above tests the difference in strength between 15 and 30-year blue gums.

I believe mining companies would do well to plant eucalyptus for their future supply of mine timbers, not necessarily at the mine, but in the nearest locality available for a commercial plantation. A careful selection of species suitable for this work and a thorough investigation of local climatic conditions, by someone familiar with the requirements and cultivation of eucalyptus, may save money and avoid failures."—L. W. Symmes.—*Mining and Scientific Press*, Dec. 26, 1908, p. 870. (W. R D.)

Blasting Gelatine. — "Considerable interest has been aroused, not only in Colorado but throughout the country, by the rapid advance recently made by Contractor A. E. Carlton in driving the Cripple Creek drainage tunnel.

In February, 1908, when Mr. Carlton took up this work, the 'double-shift' system was inaugurated, but the great difficulty in blasting the round quickly and completely made it almost impossible to drill down, fire, and throw back the muck twice in 24 hours. This was due principally to Mr. Carlton's inability to secure an explosive quick and powerful enough to bring out the 10-ft. cut hole without firing them many times, the rock, a close tough granite, offering great resistance at the points of these cut holes, which were started 2½ ft. on either side of the centre of the tunnel face and drilled so that their points almost came together on the centre line of the tunnel.

Heavy charges of 60% gelatine dynamite were used for some months in these cut holes, but without very satisfactory results, the best month's work up to September carrying the face back 301 ft. only. At this time it was decided to make a trial of Du Pont blasting gelatine, and accordingly a small quantity was secured and the cut-holes loaded with the same amount of this explosive as would have been used of 60% gelatine dynamite. The result of the first blast was terrific. Every pane of glass in the engine house and smith shop outside of the tunnel entrance and more than 3,000 ft. from the face being shattered to bits. The 24-in. ventilator pipe and 6-in. air line were blown down 500 ft. out from the face, and things mixed up generally. However, Mr. James A. McIlwee, the superintendent, immediately profited by his first experience with the powerful explosive, and after a few more trials succeeded in adjusting the charges satisfactorily by loading from four to seven 1¼ in. × 8 in. cartridges of blasting gelatine in the points of the cut holes, and six or seven 1½ in. × 8 in. cartridges of 60% gelatine on top with tamping to the face—the detonator, a quintuple or preferably a No. 20 blasting cap, being placed in the top or outside cartridge of the blasting gelatine and behind the gelatine dynamite. With the use of the blasting gelatine, Mr. McIlwee was able to bring out his cut to the back of the holes in two shots, the first shot blowing out about half of the cut and breaking back and chambering the cut holes so that a pocket was formed where the two holes came together, large enough to take from fifteen to twenty cartridges of 1½ in. × 8 in. 60% gelatine dynamite for the second shot. As this second charge could be concentrated directly back to the centre of the cut which was already relieved of at least one-half of its resistance, it never failed to bring out all of the remainder of the cut.

Occasionally one cartridge of blasting gelatine was also used in the point of each of the 'relief-cut' holes but usually a charge of 60% gelatine dynamite was all that these holes required. The remainder of the round, including side or rib, top and bottom of lifter holes, was charged with 50% gelatine dynamite.

When the tunnel was measured up at the end of September, Mr. Carlton had to his credit a drive of 353 ft. for 25 working days, or a gain over his best previous month of almost 17% in returns with practically no increase in outlay, as the slightly greater cost of the blasting gelatine over that of gelatine dynamite was offset by the reduction in quantity used." — *Mines and Minerals*, Jan., 1909, p. 282. (A. R.)

Caisson Disease.—"A commission of Belgian medical experts examined 166 caisson workers before and after their work, the shift lasting from 8 to 12 hours, and found (1) that the blood-making function, as shown by the hæmoglobin contents, was actually increased during their work; (2) that so long as the pressure does not increase beyond 3 atmospheres (45 pounds) the men feel perfectly well and perform their labour with more ease and even less fatigue than under normal atmospheric pressure; (3) that men of temperate habits, with a sound heart, lungs, and nervous system, suffer no injurious effects, and none others should be employed; (4) the real injury is done by a sudden removal of atmospheric pressure in a hasty "locking-out" process, for which the workmen are often to blame.

The general rule in "locking out" should be to allow at least one minute for each 6 pounds of pressure within the chamber.

The symptoms of the so-called caisson disease are rarely observed until the pressure equals 20 pounds, and usually do not appear for some minutes or hours after emerging. In addition to the symptoms already mentioned, there may be hemorrhage from the nose, mouth, and ears; headache, dizziness, rapid pulse, sweating, severe pain in the back, extremities, or region of the stomach, and vomiting. Partial deafness and symptoms of motor paralysis, more or less general, but most frequently confined to the lower extremities, are frequently observed. Cases with pronounced head and spinal symptoms usually prove fatal. The milder cases, as a rule, recover sooner or later, although the muscular pains and paralytic symptons may persist weeks or even longer."—*Bulletin 75, U. S. Department of Labour.—Mines and Minerals*, Jan., 1909, p. 274d. (W. R. D.)

Lubricating Ropes and Guides.—In *Glückauf* for January 2, Mr. Dobbelstein describes certain devices recently introduced into German mining practice for greasing cage ropes and guides, on account of the waste and imperfect manner in which the work is done by hand. At the Hugo colliery of the Harpener company, a hinged wrought iron casing is arranged in connection with a compressed air pipe. The rope passes through a central bore of the casing, and engine oil (which possesses the necessary fluidity and prevents rusting) is forced through a number of small orifices at an angle of 45° with the rope. The bore of the casing is eased at the top and bottom, to prevent loose ends of wire from being torn out of the rope; and the casing is mounted on rollers, so as to follow any deviations of the rope from the centre of the winding compartment. The consumption of oil is only one-sixth that of the solid grease previously used; and there is also a considerable saving in labour, one man being able to put the apparatus in position for use, whereas hand greasing takes three men a full day. Experiments made with a view to obtaining an efficient, non-slipping, anti-corrosive lubricant for underground haulage ropes showed that the best results for this purpose are obtained by a mixture of tar and driers, this possessing the requisite tenacity and elasticity. The problem of greasing cage guides is somewhat difficult, oil or grease being prohibited, on account of the danger of fire, whilst soft soap soon washes off in wet shafts, and is also wasteful in application by hand. A thin mixture of soap and graphite was eventually prepared, which greatly lessens the friction of the cage shoes on the guides, and can be applied by compressed air. For this purpose a closed vessel, divided into two compartments, for the grease and air respectively, is mounted on tub wheels, and charged with compressed air from the main until a constant pressure is indicated by the gauge. The mounted vessel is run into the cage, and a branched delivery pipe leading from the oil chamber is fitted with two pairs of nozzles that pass through holes in the side of the cage and discharge the grease on to the inner corners of the cage guides as the cage moves down the shaft, the compressed air being admitted to the oil chamber through a reducing valve. A saving of about £20 per annum is effected by the use of this device."—*Mining Journal*, Jan. 16, 1909, p. 79. (A. R)

Stresses on Winding and Conducting Ropes.—At a recent meeting of the Manchester Geological and Mining Society Mr. J. Hindley read a paper by himself and Mr. J. Stone on the above subject.

By means of an indicator which they had designed and constructed for ascertaining the stresses on winding ropes, a number of experiments had been conducted and diagrams obtained showing the stresses during winding. As a result of their experiments they concluded that the prevalent idea that the greatest stress on a winding rope occurred at the beginning of a wind was shown to be entirely wrong. In every case the application of the stress was shown to be very gradual. The greatest stresses on the winding rope were due to the vibrations of the cage. The vibrations were caused by (1) the pulsations of the engine, (2) the sudden checking of speed by shutting off the steam, or by the application of the brakes.—*Mining Engineering*, Jan. 1909. (C. B. S.)

Bending Stresses in Wire Ropes.—In a paper read before the Australasian Institute of Mining Engineers, the author criticises the Reuleaux and Hrabak formulæ for the stresses due to bending in a wire rope. From the change of curvature of a single helix whose axis is bent into a circle it is readily deducted that the maximum stress due to bending is $(Ed \cos^2 \alpha \cos^2 \beta)/D$, where E is Young's modulus for the steel of which the wires are made, d is the diameter of each wire, D that of the rope, α the angular pitch of the wires in the strand, β that of the strands in the rope. In direct tension each helix of wire is not perfectly free to behave as a spiral spring in tension; hence, although the elongation of the rope as a whole is greater than it would be if it consisted of a bundle of parallel wires, yet it is not so great as for a number of free helices. Doubtless to some degree the same action occurs in a rope bent over a pulley. Tables give the tensions required to produce the same stress as given by the bending for various ropes and diameters of drum. The results of experiments are given in which pieces of rope were bent by a known force or to a known radius. The formula is found to be in good agreement with the results. By bending in the opposite direction the work absorbed in overcoming internal friction is estimated. This amounts approximately to $Ml/2R$, where M is the bending moment required to bend a length l of the rope to radius R.—*Indian Engineering*, Jan. 16, 1909, p. 45. (A. R.)

To Destroy Explosives.—The best way to destroy ordinary black gunpowder is to throw it into a stream under conditions that prevent any harm coming to human beings or animals through the dissolving of the saltpetre. If no suitable stream is available, the gunpowder may be stirred, with water in tubs, or the dry gunpowder may be poured out on the ground in a long thin line and ignited with a fuse at one end. To destroy dynamite cartridges, the paper wrappings should be carefully removed, the bare cartridges laid in a row with their ends in contact, and the first cartridge ignited with a fuse without a cap. Even with these precautions a simultaneous explosion of the entire mass may occur, so that it is wise to retire to a safe distance. The row of cartridges should be laid parallel with the wind and ignited at the leeward end, so that the flame will be driven away from the mass. Frozen dynamite should be handled with especial care, as its combustion is peculiarly liable to assume an explosive character. A small quantity of dynamite may be destroyed by throwing it, in very small bits, into an open fire, or the cartridges may be exploded one by one in the open air with fuses and caps. Dynamite should never be thrown into water, as the nitro glycerine which it contains remains undissolved and capable of doing mischief. Other explosives which contain nitro-glycerine should be treated in the same way as dynamite. Ammonium nitrate explosives may be thrown in small fragments into an open fire, or, if they do not contain nitro-glycerine, may be destroyed by means of water. Explosive caps should be exploded singly with pieces of fuse.—*Engineer.—Science and Art of Mining*, Jan. 23, 1909, p. 266. (A. R.)

BLOWN-OUT SHOTS.— 'A blown-out shot may thus be defined: (1) 'A shot insufficiently stemmed whereby the tamping offers less resistance than the force generated by the combustion of the explosive.' So that the rocks in which the charge is inserted do not collapse or shatter, but allow the explosive force to escape through the narrow shot hole at a tremendous velocity causing the flame and force of explosion to extend a considerable distance, at the same time generating great temperature to such an extent that, though in ordinary conditions the flame of certain explosives is not of sufficient temperature to ignite gas and dust, it will, subject to enormous velocity and explosive force, when coming in contact with the small particles of fine dry coal dust, ignite them. This is illustrated by stones falling from some height and striking another blow, and the force of contact generating a spark. The concussion of the air current in the vicinity of a heavy blown-out shot is sufficient, in my opinion, to ignite the small particles of coal dust, when we remember that each particle is surrounded with a thin envelope of highly inflammable mixture called 'ferrous sulphide' caused by decomposition, which becomes more prominent after the dust is treated by water and has since dried; and again during the drying process of the dust it has been proved that the vapour arising from the dust is nearly as inflammable as petroleum. This vapour, subject to severe concussion, can have sufficient temperature imparted to it to ignite the coal dust, and form the first fruits of a terrible disaster.

(2) A blown-out shot may be defined as an overcharged shot-hole, whereby too much explosive is introduced, hurling the rocks a long distance, at the same time a portion of the flame escapes by way of the shot-hole with tremendous force, often greater than the first instance, and with the greater liability of causing an explosion of coal dust. Also, the flying stones may seriously injure persons who may not have been properly sheltered themselves, and, again, the flying stones may knock timbers out, causing a large fall, and an outburst of fire-damp, which may become ignited by the falling stones, or the flame of the blown-out shot, should the fall occur practically instantaneously, as we have records of explosions caused in this manner.

(3) A blown-out shot may be defined as a shot unskilfully placed, or in a crack or thin layer of coal. Such shot may have been properly stemmed, and with a proper charge of explosive, yet the products of combustion escape by way of the cracks in the rocks, or above or below the thin layer of coal, especially if the collier allows the rippings to hang some distance, which may have loosened sufficiently to cause a thin cavity, often unseen, which at the same time may be full of fire-damp impossible to detect. It can be readily seen that a shot fired under these circumstances is dangerous in two ways, by blowing out and igniting dust, and at the same time igniting the fire-damp concealed in the rippings, causing a serious accident, setting fire to everything inflammable within its reach, which may extend with terrible rapidity and spread and create the horrible disasters we are all too well acquainted with. Again, the heat generated by shots when spreading among cracks in the rocks may be sufficient to ignite other explosives which may yet remain to be exploded by the shot-man, and such explosion may occur just as he is in the act of lighting the fuse or connecting the cable to the detonator wires, or possibly inserting explosives in uncharged shot-holes. Many men have lost their lives in this way. A case similar to this occurred after a blown-out shot at Glyncorrwg Colliery, South Wales, where a shot-man attempted to insert a charge in a hole that had blown out in a hard heading, and there was sufficient temperature in the shot-hole to ignite the second charge, instantly killing the two men who were engaged on the work. At the inquest it was said that even cold water had been splashed into the shot-hole before re-charging, yet the charge fired, proving the great heat generated during blown-out shots.

(4) A blown-out shot may also be defined as one where the charge is inserted, and before being tamped has ignited by the heat generated by previous shots, blowing out with tremendous force, according to the amount of explosive that has been inserted, and its density.

(5) A blown-out shot may also be defined as one where a shot fires prematurely by pressing a charge into a hole of insufficient size, or by using an iron bar, etc.

(6) A blown-out shot is also one caused where the hole is not properly cleaned, and a thin layer of stone or coal dust remains on the sides of the hole, which may allow the products of combustion to escape even after being tamped.

Precautions to prevent Blown-out Shots:—

(1) The colliery manager to comply with the C.M.R. Act by appointing the most skilful and careful shot-men he can find, and give them a short lecture as to their duties, the dangers of shot-firing, explaining how accidents may occur, and testing the knowledge of the shot-men in the general practical requirements, also their knowledge of general rule 12 and its amendments. No man should be appointed a shot-man who does not know general rule 12.

(2) Instruct the shot-man to charge and fire all shots himself, and not to fire more than a specified number, as the manager may deem safe. In our colliery only five shots are allowed to be fired in each district each shift.

(3) Each collier to state the length of each shot-hole with chalk on a piece of board, so that the shot-man can properly proportion the charge required.

(4) The shot-man should properly stem the shot-hole from the charge to the mouth of the shot-hole.

(5) He must not fire shots placed in cracks or thin layers of coal.

(6) He must not stem the hole with coal dust, but only good plastic clay, or tamping plugs.

(7) Do not press a charge into a hole of insufficent size, or tamp the first portion too hard on the charge, and only use a proper tamping rod of copper or wood.

(8) Properly clean out every shot-hole of dust with a copper scraper before inserting the charge.

(9) The heading or coal face should be properly undercut or holed, and see that all unnecessary resistances are picked away and removed as sprays, and pieces of coal or stone.

(10) Shot-holes should not be too short, and should be of sufficient diameter to only just allow the explosive to pass easily.

(11) Where there are high currents of air the shot-holes should be placed with the direction of the air current with the mouth of the shot-hole pointing from the downcast shaft. Thus the shot will not blow against the air. No doubt explosions have been caused in this way, due to the concussion caused by the explosion of the shot against the air and dust which contains occluded gas, or heavy hydro-carbons, which according to Prof. Bedson, are even more readily explosive than fire-damp, and take fire at a lower temperature.

(12) Prohibit gunpowder, and use safety explosives as a substitute to secure greater safety in dusty and gaseous mines where blasting may be necessary.

(13) See that the dust and all parts of the roadway are thoroughly wet for a distance of 20 yards radius of the shot-hole to prevent the extension of fire should a blown-out shot occur.

(14) Comply with the C.M.R. Act strictly in case of the presence of fire-damp; as before stated, dust alone is highly dangerous, but becomes more so when assisted by fire-damp.—DAVID FOSTER.—*Science and Art of Mining*, Jan, 23, 1909, p. 281. (A. R.)

MISCELLANEOUS.

CAPACITY OF PIPES.—"This table shows the number of gallons discharged per minute through pipes for specified sizes and grades.

—	Fall per 100 ft.							
Size of Pipe.	1 in.	2 in.	3 in.	6 in.	9 in.	1 ft.	2 ft.	3 ft.
3 in. ...	13	19	23	32	40	46	64	79
4 in. ...	27	38	47	66	81	93	131	163
6 in. ...	75	105	129	183	224	258	364	450
8 in. ...	153	216	265	375	460	529	750	923
9 in. ...	205	290	355	503	617	711	1006	1240
10 in. ...	267	378	463	655	803	926	1310	1613
12 in. ...	422	596	730	1033	1273	1467	2076	2554
15 in. ...	740	1021	1282	1818	2224	2464	3617	4467
18 in. ...	1168	1651	2022	2860	3508	4045	5704	7047
24 in. ...	2396	3387	4152	5871	7202	8303	11744	14466
30 in. ...	4187	5920	7252	10257	12580	14504	20516	25277

Canadian Engineer.—The New Zealand Mines Record, Dec. 16, 1908, p. 233. (A. R.)

ACTION OF GASES ON BOILER PLATES.—"It is well know that even the best Lowmoor furnace plates have, after years of work, acquired a brittleness which cannot then be removed by annealing, and Mr. M. Longridge has recently published details of a boiler which had been in continuous and satisfactory work for 70 years, but which had become so brittle that holes could be knocked into it with a hammer. Such plates would, of course, not stand renewed manipulations in the boiler shop, and might easily crack if repairs were carried out, but if left undisturbed they are likely to do their duty in resisting ordinary stresses, because their strength has not been diminished; on the contrary, it may have been increased. The cause of this brittleness may be either a molecular change due to age alone, or a chemical change due to absorption of gases. It is not as widely known as it should be that neither solids nor fluids are free from gases, that it is most difficult to remove them, and that absorption is constantly taking place. Glass is by most people looked upon as being impervious to gases, but the gradual change of soaped glasses to an amethyst tinge, which is sometimes very noticeable with windows of old houses, proves not only that these glasses absorb oxygen, but also that the oxygen effects a chemical change of the manganese salts in the glass. It is, therefore, not surprising that the noxious fumes of boiler furnaces enter the plates, and possibly also hydrogen from the water side. It is, at any rate, certain that a very small percentage of hydrogen makes mild steel not only brittle, but so hard that it will scratch glass, and very small traces of nitrogen also impart brittleness to mild steel. Unfortunately, these gases are not easily detected by chemists, and are rarely recorded in their analyses, so that when cases of brittlenes are dealt with the possibility of its being caused by gases cannot be ascertained."—C. E. STROMEYER, Manchester Steam Users Association.—*Page's Weekly*, Jan. 1, 1909, p. 2. (A. R.)

CONVERSION TABLES FOR ASSAY VALUATIONS.—The following tables, or rather portions of them, were computed some years ago to facilitate the reduction of certain assay values to a common basis. They have been extended as far as seems desirable, and are presented to the readers of *Mines and Minerals* in the hope of saving time and labor to some of them.

Bases of Computation.—The gram is taken as 15,4320 grains. The value of a troy ounce of fine gold is assumed as being exactly $20.67, instead of $20 6718346+, resulting in an error of less than one in 10,000. Values in English coin are based on the assumption that an ounce of fine gold is worth 4·25 pounds sterling, or 85 shillings, 1,020 pence; this is too high by about one part in 2,000, the true value being 1,019.45 pence. It is useless to attempt a closer approximation in practical work, for the simple reason that gold bullion assays are rarely reported closer than the nearest half millieme, or to within one part in 2,000. At the values adopted one dollar is equivalent to 4.11224 shillings, and one pound sterling to $4.86353.

Foreign and Obsolete Values.—The adarme (36 grams or $\frac{1}{16}$ of the Spanish ounce), sometimes used by Mexicans colloquially, and especially with reference to placer work, is about 1.8 grams, which in fine gold would be worth $1.20. For practical purposes, however, an adarme of ordinary gold may be taken as equivalent to $1, and this exactly true for gold 830 fine.

Russian reports state values in zolotniks per 100 poods, but for low-grade placer deposits doli per 100 poods are used. As a dola is $\frac{1}{96}$ zolotnik we may take $\frac{1}{100}$ of the values given in the table for the zolotnik, without serious error.

Marcos per cajon were formerly used in some South American countries, one marco per cajon apparently ranging between 100 and 70 parts per million. One oitavo per quintal corresponds practically to 2 ounces per ton.

The loth per centner, used in the older German works, corresponds to one part in 3,200, which is nearly 9 ounces per short ton or 10 ounces per long ton. In some cases there seems to have been a considerable variation from this ratio, the value being sometimes taken as one part in 3,520, or $\frac{10}{11}$ of that used in these tables—the centner being then assumed as 110 instead of 100 pounds. The quentchen was $\frac{1}{4}$, and the denär $\frac{1}{16}$ of the loth."—W. J. SHARWOOD.—*Mines and Minerals*, Jan., 1909, p. 250. (W. R. D.)

FINENESS OF BULLION AND ALLOYS OF PRECIOUS METALS.

Denomination.	Equivalent in Millièmes or parts per Thousand.	
1 carat ...	41·660	24 carats = 1 lb. troy (England) ,, 1 mark (Germany, etc.)
1 gr. per marc	·217	4,608 gr. = 1 marc of 8 oz. (France, Spain, etc.)
1 oz. per marc	125·000	8 oz. = 1 marc (France, Spain, etc.)
1 loth (silver)	62·500	16 loth = 1 mark (Germany etc.)

VOLUME AND WEIGHT OF FINE GOLD AND SILVER.

	One Cubic Centimeter	One Cubic Inch.	One Cubic Foot.
Fine Silver:			
Weight: grams ...	10·57	173·21	299307·00
Weight: troy ounces	·339825	5·5687	9622·72
Fine Gold:			
Weight: grams ...	19·3	316·269	546,513
Weight: troy ounces	·6205	10·1680	17,570·39
Value: U.S. dollars	$12·82^{57}	$210·17	$363,180
Value: pounds sterling	£2·647	£43·214	£74,674

ASSAY VALUATIONS.

Values.	One Part In.	Per Cent.	Per Metric Ton.			Per Long Ton of 2,240 Pounds.			Per Short Ton of 2,000 Pounds.	
			Troy Ounces.	U.S. Dollars.	Grams.	Troy Ounces.	Oz. Dwt. Gr.	U.S. Dollars.	Troy Ounces.	U.S. Dollars.
1 per cent. ...	100	1	321·50	6,645·40^{6}	10,000	326·666	326 13 8	6,752·20	291·666	6,028·75
1 gm. per metric ton = one part per million ...	1,000,000	·0001	·03215	·66^{45}	1	·03266	15·68	·67^{5}	·029166	·60^{29}
1 troy oz. per short ton ...	29,166·66	·00342857 = $\frac{24}{7000}$	1·1023	22·78^{45}	34·2857	1·120	1 2 9·6	23·15	1	20·67
1 troy oz. per long ton ...	32,666·66	·0030612 = $\frac{3}{980}$	·9842	20·34^{34}	30·612	1	1	20·67	·892857	18·45^{8}
1 troy oz. per metric ton ...	32,150	·0031104 = $\frac{2}{643}$	1	20·67	31·104	1·016	1 7·7	21·00	·90720	18·75^{2}
1 dollar gold per short ton ...	602,875	·00016587	·0533285	1·1023	1·6587	·054185	1 2	1·12	·048379	1
1 zolotnik per 100 poods ...	384,000	·0002604	·083724	1·73^{06}	2·604	·08507	1 16·83	1·75^{8}	·075954	1·57
1 loth per centner	3,200	·03125	10·046875	207·66	312·5	10·2183	10 4 9	211·21	9·11458	188·14
1 oitavo per quintal ...	16,384	·00610	1·9623	40·56	61	1·9938	1 19 21	41·21^{18}	1·78654	36·80

CONVERSION TABLES—WEIGHTS.

	Grains.	Pennyweights.	Troy Ounces.	Avoirdupois Ounces.	Avoirdupois Pounds.	Grams.	Fine Gold Value.	
							United States.	British.
1 Grain ...	1	·041666	·0020833	·00228571	·000142857	·0648	4·306c.	2·125d.
1 Pennyweight	24	1	·0500	·0548571	00342857	1·5552	$1·0335	4·25s.
1 Troy ounce ...	480	20	1	1·0971428	·0685714	31·104	$20·67	85s.
1 Troy pound ...	5,760	240	12	13·165714	·822857	373·248	$248·04	£51
1 Avoirdupois oz.	437·50	18·22917	·911458	1	·06250	28·35	$18·84	77·474s.
1 Avoirdupois lb.	7,000	291·666	14·58333	16	1	453·60	$301·4375	£61·97
1 Milligram ...	·015432	·000643	·00003215	·000035274	·0000022046	·0010	·06645c.	·033d.
1 Gram ...	15·432	·643	·03215	·035274	·0022046	1	66·45c.	2·73275s
1 Kilogram ..	15,432	643	32·15	35·274	2·2046	1,000	$664·54	£136·64

THE MICROGRAPHICAL STUDY OF CEMENT.—In view of the advantages which have followed the micrographical examination of metals, the Prussian Royal Laboratory for the Testing of Materials have recently, according to the *Builder*, applied the same method to the study of various cements and in particular to Portland cement. By polishing the surface of Portland cement clinker the investigators were enabled to identify the two principal constituents—described as *alith* and *belith*—which were found to occupy unequal and unequally distributed areas of the surface under examination. The application of re-agents such as hydrochloric, nitric, acetic, salicylic and iodic acid showed still more clearly the differences between these two constituents. Among other re-agents a weak solution of citric acid violently attacked the polished surface, which was quite unaffected by oxalic acid. Hydrofluoric acid gave a characteristic reaction, revealing by discoloration the presence of slag in some specimens. By heating samples to high temperatures the investigators were enabled to ascertain that the hardest crystals—those of *alith*—were not a simple hydrate of calcium, since they were able to stand without disintegration the temperature of 900° C. Finally, the examination of thin laminæ of cement served to illustrate the optical properties of these crystals and revealed the presence of a third constituent—described as *celith*. So far, however, the research has thrown no light upon the nature of the re-action or molecular re-arrangement responsible for the setting and hardening of Portland cement.—*Indian Engineering*, Jan. 16, 1909, p. 45. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

THE ORE DEPOSITS OF SOUTH AFRICA. PART I. Base Metals. By J. P. JOHNSON. 5s. Net. (London: Crosby, Lockwood & Son.)

That there should have been no handbook of reference written on South African economic geology since Wilson-Moore's little volume appearing fifteen years ago is by no means creditable to local geologists, but it is a circumstance that should be of considerable significance in giving Mr. J. P. Johnson's treatise, named above, the circulation it merits. Having so clear a field Mr. Johnson was able to deal with his subject from any standpoint. He has worked on thoroughly scientific lines and if, in consequence, he has limited the range of popularity, he has at least produced a work of more enduring value. The book may be said to form a good companion work to Hatch & Corstorphine's "Geology of South Africa." The claim made for it is that it should meet a demand "for a co-ordinated and condensed account of the ore-deposits at present known in South Africa"—the part issued only covering the base metals. Considering the brevity of the production, this aim has been skilfully achieved. In thirteen chapters running to sixty pages in all, the author covers a remarkably wide field. The first chapter is somewhat ambitious for a volume of the present scope and treats with risky conciseness of the origin and classification of ore deposits. Those who have followed the famous controversy of the "Ascensionists" and their antagonists, excited by Posepny's classic paper on the "Genesis of Ore Deposits" will sympathise with Mr. Johnson in his efforts to present an eight-page condensation of the subject as a guide to prospectors, and, at the same time, commend the measure of his success.

The body of the volume is made up of short accounts of the occurrences in South Africa of the following metalliferous ores: titaniferous and chromiferous iron oxides, nickel, copper, cobalt, tin, molybdenum, tungsten, lead, mercury, antimony and iron. Descriptions are based for the most part on personal observation, but other authorities are frequently quoted and fully acknowledged. As a broad review, these chapters are excellent and while the student of any particular class of product may find the data too scanty, he will generally be put on the track of more detailed references. The treatment of the subject should appeal to many readers in Europe and America, who are anxious to learn at a glance the present stage of South African base metal mining development.

The final chapters comprises "Hints to Prospectors." Mr. Johnson practically states that there are no competent prospectors in the country. It is to be feared, however, that the standard of qualifications he establishes is rather elevated and would bar the claims to recognition held by the host of "prospectors" who by virtue of their numbers rather than an individual knowledge of "magmatic differentation" or "metasomatic replacement," add yearly to the world's wealth by their discoveries in Australasia or America. Mr. Johnson is so zealous a supporter of the genuinely scientific side of his profession that he maintains a higher standard of thought and expression than that commonly met with in handbooks for the prospector's guidance. In the first place, he presupposes a better grounding in the elements of geology than the average prospector in any land possesses; in the second, he does not give those instructions as to field equipment and simple expediencies for the rapid determination of minerals, which usually appear in the popular vade mecum.

If the defects of Mr. Johnson's volume have been discussed at disproportionate length and its signal merits too briefly noticed some excuse may be found in that a second part is to follow and omissions can be simply remedied. An index, at least of place-names, would be serviceable. If Mr. Johnson should later produce a volume on the precious metals, and a third on precious stones and such mineral products as asbestos, lime, mica, graphite and magnesite, etc., he would fill a deplorable blank in South African scientific literature, and, judging by his first publication, would fill it well. (R. S.)

COAL MINING. By D. BURNS and G. L. KERR. Part III. (London: Whittaker & Co., 2, White Hart Street, Paternoster Square, E.C.)

"The whole of Part III. of this publication is devoted to the sinking and fitting of shafts, a subject of primary importance, both to the mining engineer and colliery owner. The general character of the work as a comprehensive and practical guide on the technical side of coal mining is admirably sustained in this section. The treatment of the subject is exhaustive without being discursive, while a word of special praise is due to the illustrations, which are not only numerous but of so eminently practical a type as greatly to assist the task of the student. Commencing with the forms of shafts, the authors point out the advantages and disadvantages of the circular and rectangular types, and proceed with a series of calculations for determining the number and size and position of the shafts required. The sinking of circular shafts is then described and illustrated in great detail, the respective merits of brick walling and concrete lining being touched upon and full directions given for the work of construction. Not the least valuable feature of this portion of the book is the information given relative to the cost of sinking, not only in this but in foreign countries also. The water problem receives full consideration, and the calculations involved in connection with shaft tubbing are presented with commendable clearness. The raising of the débris and its handling on the surface are also subjects upon which the authors have much useful information to convey. With the same thoroughness and grasp of the subject the sinking of rectangular shafts is then proceeded with, followed by a section upon the enlarging and deepening of shafts. The different methods adopted in sinking through running sand or mud are fully dealt with, prominence being given to the compressed air and freezing system. Some useful hints on the ventilation and lighting of sinking shafts, and on the best forms of bucket to employ conclude the Part, an appendix setting forth specimens of sinking specifications."—*Iron and Coal Trades Review,* Dec. 4, 1908, p. 2,418. (A. R.)

METALLIC ALLOYS: Their Structure and Constitution. By G. H. GULLIVER, B.Sc., F.R.S.E., with 104 illustrations, $8 \times 5\frac{1}{2}$, xvi. + 254 pp. (Charles Griffin &Co., Ltd.), 1908.

"The author is to be congratulated on the success of a resolute attempt to introduce a certain amount of system into a subject which has hitherto consisted rather of a series of disconnected experiments and investigations by various authorities than any intelligible scientific theory, treating of allows as a whole. The modern methods of investigation, by means of the etching of polished surfaces and the employment of micro-photography, may still be said to be somewhat in their infancy. In the case of many industrial alloys of commercial importance, the composition of which has long been known, we rely greatly on rule of thumb methods, but owing to the efforts of the Alloys Research Committee of the Institution of Mechanical Engineers, much light has recently been thrown on matters which have hitherto appeared obscure. The employment of the pyrometer and the study of critical temperatures in regard to certain alloys has opened up new methods of investigation, which have yielded important results, and placed in the hands of the metallurgist the means of determining facts which can be explained by no other means. In the case of certain refractory substances, the adoption of the electric furnace as an aid to research has proved of the utmost value.

The author deals fully with binary and ternary alloys, and discusses the known cases in which evidences are found of the formation of definite chemical compounds. A chapter is devoted to the equilibrium conditions in metallic mixtures with special reference to the phase rule. In the opening chapter a good outline is presented of the preparation of specimens for microscopic examination and of the mode of conducting the various investigations. This will prove a useful and suggestive work for the student, and it covers a field on which much still remains to be accomplished. The author's explanatory diagrams are clear and good, and the work is well illustrated with micro-photographs of the etched surfaces of characteristic alloys."—*The Times Engineering Supplement*, Dec. 9, 1908, p. 20. (W. A. C.)

THE MECHANICAL ENGINEERING OF COLLIERIES. Vol. II. By T. CAMPBELL FUTERS. Published for the Proprietors, The *Colliery Guardian Company*, Limited, by the Chichester Press, Holborn, London, E.C.

"The Volume before us deals with Haulage, Pumping, The Generation and Transmission of Power, Coke Ovens, and in the concluding chapter general information on such subjects as Ventilation, Coal-Cutting Machines, Coal Face Conveyors, Workshops, Brickworks, Stores, Ambulance, etc., is given in a clear and interesting manner. Throughout this work the illustrations are of a high standard, the author evidently recognising the material assistance of careful draughtmanship. There is abundant evidence of systematic arrangement, and the reader will find his progress through the work unattended with those annoying repetitions and "doublings" which go so far to mar many text-books devoted to mechanical engineering. The chapter: The Generation and Transmission of Power is in need of expansion and revision. The treatment of the alternating current motor is cursory. The volume displays careful classification, and will prove eminently entitled to a prominent place on the bookshelf of the colliery manager."—*Science and Art of Mining*, Feb. 6, 1909, p. 297. (A. R.)

AUSTRALIAN MINING AND ENGINEERING REVIEW contains a variety of interesting mining news, district reports and special articles. The "Review" is well illustrated and deals in a thorough manner with such wide subjects as Mine Surveying, Ventilation of Mines, Coal Examinations, and Australian Railways, and a special section is devoted to Electricity; it should have a useful and prosperous career, filling a much felt want. (C. B. S.)

THE ELECTRICAL EQUIPMENT OF COLLIERIES.—By W. G. DUNCAN and D. PENMAN. (London: Messrs. Scott, Greenwood and Son). Price, 10/6 net.

"The opening chapters call for little or no comment. General principles, magnetism, units, dynamos, and motors are described and illustrated, and in a manner similar to that of the usual text-book. In the chapters dealing with the Transmission and Distribution of Power, Prime Movers, and Lighting by Electricity much useful information is given. "Initial outlay and Working costs of Electrical Installations" are really good, but when the authors come to electricity in practical use in the mine, viz., coal-cutting, hauling, winding, pumping, drilling, we follow them with respect. Here they are on their own ground. The information, data, etc., incorporated is worth the price the publishers ask for the book. The subject of shaft signalling is meagrely dealt with. The chapters on Dynamos and Motors and Power Transmission could be extended with advantage to include some more illustrations and full description—preferably diagrammatic—of modern circuit breakers, discriminating apparatus, networks, synchronisers, etc. Producer gas plant merits their observation. Would they have their readers know the value of the apparently valueless refuse of the mine?

The publishers are deserving of praise. The real book-lover will endorse our statement that printing, paper, binding, etc., are of a high standard.—*Science and Art of Mining*, Feb. 6, 1909, p. 297. (A. R.)

Raffety, Charles W. An Introduction to the Science of Radio-activity. Cr. 8vo., pp. 220. *Longmans.* Net 4s. 6d.

Clayton, Edwy Godwin. A compendium of Food-Microscopy, with section on Drugs, Water and Tobacco. Illustrated. 8vo. pp. 468. *Ballière.* Net 10s. 6d.

Badgley, Colonel W. F. 'Heat and other Forces. Part 1, Heat. Part 2, Physical Force. Crn. 8vo. pp. 221. *King, Sell and Olding.* Net 1s.

Clowes, F. and Coleman, J. B. Elementary Practical Chemistry. Part 2, Analytical Chemistry, Qualitative and Quantitative. 6th edition. Cr. 8vo. pp. 258. *Churchill.* Net 3s. 6d.

Keane, Charkes, Alexander. Modern and Organic Chemistry. Crn. 8vo. pp. 529. *W. Scott* 6s.

Tilden, W. A. Chemistry (Dent's Scientific Primers.) Net 1s.

Handbook to the Technical and Art Schools and Colleges of the United Kingdom. Compiled from Official Information. With an Index to Courses of Instruction. 8vo. pp. 152. *Scott, Greenwood.* Net 3s. 6d.

Japtner, H. V. Heat Energies and Fuels. 8vo. *Spon.* Net 12s. 6d.

Parsons, J. Jones. Malleable Cast Iron. 8vo. pp. 184. *Constable.* Net 8s.

Prelini, C. Graphical Determination of Earth Slopes, Retaining Walls, and Dams. 8vo. *Constable.* Net 8s.

Rosenthal, L. W. Practical Calculation of Transmission Lines. 8vo. *Spon.* Net 8s. 6d.

Mineral Kingdom, The. By Dr. Reinhard Brauns. Trans. with additions by L. J. Spencer. With 91 plates (73 of which are coloured) and 275 text figures. Part 1. 4to sd. *Williams and Norgate.* Net 2s.

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 85/09. Fraser Alexander. Improvements in means for diverting the pulp from one amalgamating plate on to an adjoining plate. 24.2.09.

(C.) 87/09. Guiseppe Nicosia. An improved artificial stone and the process of manufacture thereof. 25.2.09.

(P.) 88/09. Herbert Lewis Burgess (1), William Phillip (2). Controlling means for steam traps. 25.2.09.

(C.) 89/09. William D'Arcy Lloyd (1), Ernest Thomas Rand (2). Improvements in apparatus for extracting precious and other metals from solution. 26.2.09.

(C.) 90/09. John Collins Clancy. Treatment of ores bearing precious metals. 26.2.09.

(P.) 91/09. John Cochran Dann. Improvements in couplings for railway vehicles. 26.2.09.

(C.) 92/09. William Vandyke Watson. Process for the extraction of precious metals from ores. 26.2.09.

(C.) 93/09. William Vandyke Watson. Process for the precipitation of precious metals from cyanide solutions containing the same. 26.2.09.

(C.) 94/09. Gesellschaft für künstlichen Zug, Gesellschaft mit beschränkter Haft (1), Georg Schwabach (2). Method and means of conveying gases. 26.2.09.

(P.) 95/09. Basil Robertson Lusk. Improvements in the treatment of antimony-bearing ores and the like. 26.2.09.

(P.) 96/09. Charles Henry Perrins. A new and improved bit or cutter for rock drills. 27.2.09.

(C.) 97/09. George Arthur Pearson. Improvements relating to couplings for air and vacuum brake pipes. 27.2.09.

(P.) 98/09. George Arthur Pearson (1), John Bernard Thompson (2). Improvements in mine trucks and the like. 1.3.09.

(P.) 99/09. Bruce Edward Tennant. Improvements in assay furnaces. 1.3.09.

(C.) 100/09. Ernst Otto Schneider. Improvements in cylindrical positive electrodes. 1.3.09.

(P.) 101/09. Harry Olver. Improvements in means or devices for receiving and issuing tokens or checks against deliveries of drills, tools and other appliances employed in mining and other occupations. 2.3.09.

(P.) 102/09. Harry Olver (1), Nicholas Hocking (2). Improvements in clips or devices for bundling rock drills and the like. 2.3.09.

(P.) 103/09. Rollo Bowman Ballantine. An improved process for the reduction of iron oxides, and apparatus therefor. 3.3.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices

ANDERSON, W., *l/o* Cleveland; Antelope Mine, Malopos, Rhodesia.

ANDERSON, A. G., *l/o* Rezende; Witwatersrand Deep, Ltd, P. O. Box 5, Knights.

BRISTOL, J. J., *l/o* Germiston; Reno Nevada, U.S.A.

CHEW, B., *l/o* Johannesburg; Moodie's Fortuna G. M. Co., Ltd., P. O. Box 124, Barberton.

COE, W. C., *l/o* Lomagundi; Rowdy Boys Mine, near Eldorado, S. Rhodesia.

COGLE, C. T., *l/o* Fordsburg; P. O. Box 531, Johan-

CREED, J. P., *to* 157a, Church Street, Turffontein, Johannesburg.

nesburg.

EDWARDS, A. L., *l/o* Barberton; P. O. Box 727, Johannesburg.

ELMORE, F. E., *to* 72, Gloucester Terrace, Hyde Park, London, W.

GAZZAM, J. P., *to* 5027, Westminster Place, St. Louis, Mo.. U.S.A.

GERARDY, B. G., *l/o* Cleveland; Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston.

HALLIMOND, W. T., *l/o* Fordsburg; Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland.

HAMILL, J. H., *l/o* Johannesburg; Treasury G. M., Ltd., Cleveland.

HEBBARD, A., *to* P. O. Box 58, Langlaagte.

JONES, W. A., *l/o* Selukwe; Gaika Mine, Queque, Rhodesia.

LUDLOW, L., *l/o* Bulawayo; c/o Standard Bank of S.A., 10, Clements Lane, London, E.C.

MCCRACKEN, J., *to* Sandycroft Foundry Co., Ltd., P. O. Box 1976, Johannesburg.

MELVILL, GEO., *l/o* Fordsburg; French Rand G. M. Co., Ltd., P. O. Box 25, Luipaardsvlei.

NEWTON, S., *l/o* Fordsburg; Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg.

ROBINSON, R. W., *l/o* Johannesburg; Penhalonga, Rhodesia.

WAY, E. J., *l/o* Benoni; P. O. Box 2927, Johannesburg.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. IX. APRIL, 1909. No. 10.

Proceedings
AT
Ordinary General Meeting, April 21, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, April 21st, Mr. R G. Bevington (President), in the chair. There were also present:—

47 Members: Dr. J. Moir, Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, K. L. Graham, A. Richardson, G. O. Smart, J. E. Thomas, A. Whitby, H. A. White, Prof. J. A. Wilkinson, W. A. Caldecott, Prof. J. Yates, S. Beaton, W. Beaver, W. Bradford, F. T. Chapman, A. A. Coaton, W. M. Coulter, A. D. Gilmour, G. Goodwin, J. Gray, A. Heymann, J. I'Ons, E. J. Laschinger. G. A. Lawson, Jas. Lea, G. Melvill, H. Meyer, J. E. Metcalf, P. T. Morrisby, S. Newton, T. T. Nichol, D. J. Pepler, L. J. Robinson, O. D. Ross, W. Sharp, S. Shlom, H. Taylor, A. Thomas, K. Tonnesen, O. Tonnesen, J. F. Walker, F. W. Watson, and E. M. Weston.

10 Associates: Messrs. J. Chilton, C. T. Cogle, M. Dennehy, D. Dowling, J. Ingram, A. N. Locke, H. B. Powter, Hy. Rusden, H. Stadler, and W. A. C. Tayler.

5 Students: Messrs. G. J. V. Clarence, C. S. van Hasselt, J. Innes, C. Lipschitz and G. Little.

15 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the March *Journal*, were confirmed.

NEW MEMBERS.

Messrs. Jas. Lea and F. W. Watson were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

CORDER, CECIL ARTHUR, East Rand Proprietary Mines, Ltd., P.O. Box 152, East Rand. Mining Engineer.
LOCKER, JAMES THOMAS, Warrington, England. Screening Manufacturer.
HOWARD, HAROLD ARTHUR LIDDON, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.
MURDOCH, DAVID, Simmer and Jack Proprietary Mines, Ltd., P.O. Box 192, Germiston. Cyanider.
ROSS, DAVID GEORGE FRASER, Simmer and Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Cyanider.

The Secretary: Since the last meeting of the Society the following associates have been admitted:—

BROWNE, CLAUDE MELVILLE, Cinderella Deep, Ltd., P. O. Box 75, Boksburg. Cyanider. *(Transfer from Student Roll).*
GREIG, DOUGLAS WILLIAM, Dynamite Factory, Modderfontein. Technical Chemist.
HOPKINS, JOSEPH HUXLEY, Consolidated Rand Brick, Pottery, and Lime Co., Ltd., P. O. Box 609, Johannesburg. Business Manager.
MASTERSON, CHARLES WILLIAM, Subeni Copper Mine, Vryheid, Natal. Assayer and Surveyor.
ROSS, ARTHUR WILLIAM, P. O Box 9, Day Dawn, Western Australia. Mining Engineer.

And as students:—

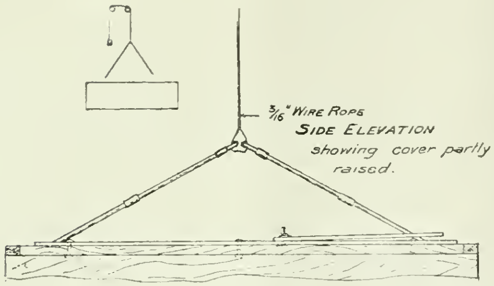
HASSELT, CAREL LOUIS VAN, College House, 42, Berea Road, Bertrams, Johannesburg.
LITTLE, GEORGE HEARD, Transvaal University College, P. O. Box 1176, Johannesburg.

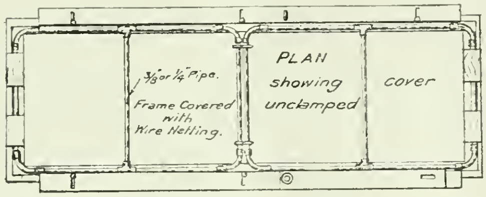
GENERAL BUSINESS.

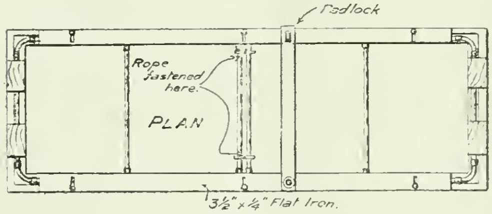
The President drew attention to the Annual Dinner of the Society taking place on the following Saturday, and expressed the hope that as many members as possible would attend the gathering.

COVERS FOR EXTRACTOR BOXES.

Mr. K. L. Graham (*Member of Council*): This sketch illustrates a neat cover for extractor boxes or mill plates, complying with the requirements of the new Gold Law. It is made to the design of Mr. W. B. Shed, of the Geldenhuis Deep Co.'s cyanide staff, and has been adopted for use there. The cost per 18 ft. box is about £6; and corrugated netting of $\frac{3}{4}$ in. holes, 18 S.W.G. is recommended for the frames. This can be imported from home, made to the right size and ready for putting on. When raised clear, the frames suspended from two points at the hinged







centre, can be raised easily by means of the counterpoise to any convenient distance above the extractor box or mill plate.

The President: We must thank Mr. Graham for having brought this little idea before us. It is a matter of great interest just now, and any suggestions of this sort are valuable. Do I understand that the cover folds?

Mr. K. L. Graham: Yes, it folds downwards. Very nearly the same thing is in use on the Crown Deep.

Mr. Jas. E. Thomas (*Member of Council*): Has anything been said so far as to which boxes are to be covered up, or as to the particular compartments in each box?

The President: So far as I know, nothing has been said beyond what the Government Mining Engineer stated at our last meeting. It would appear from his remarks that the matter is to be referred to the Mining Inspector of each district for him to inspect the boxes and say which are to be covered up and which are not. That, I take it, is the gist of the Government Mining Engineer's remarks. He distinctly stated that it was a matter to be left to the Mining Inspectors.

Mr. K. L. Graham: The Mining Inspector of the Germiston district circularised the majority of the mines, and said we had to cover the boxes by the 1st of April. We had no option in the matter. The Government Mining Engineer's remarks came rather late, and did not save us any expense at all.

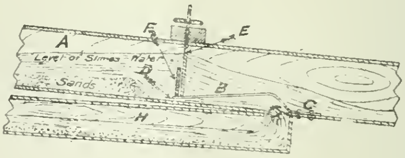
The President: That is so. In fact I think the whole country was circularised to that effect. The regulations were to be strictly enforced on and after the 1st of April.

A LOW FALL SANDS RETURN CONTRIVANCE.

Mr. W. R. Dowling (*Member of Council*) submitted the following notes on behalf of the author,

Mr. C. W. Dowsett (*Associate*): The first sketch represents a device lately put into use on the Knight Central cyanide works. It can be used where there is not sufficient fall for the sands from the bottom of a sands return box to run back to the tailings wheel or pump. The cost is only nominal, and it works well.

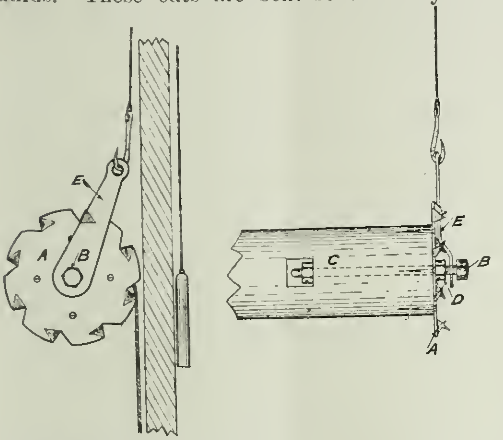
The following is a description of Sketch I.:—A, launder from sands collectors to slimes plant; B, plate of $\frac{1}{8}$ in. iron going right across launder, fitting tight at sides and bolted down to the bottom of the launder at C on a $\frac{1}{2}$ in. rubber joint. B is made so that it comes to within $\frac{1}{4}$ in. of the bottom of the launder at D: E, hand screw passing through a bearer, to raise or depress the plate at D; F, baffle going right across launder, resting on top of the end of the plate, fitting into slots at the sides of the launder, and with blanketing on the lower edge to prevent sands from escaping between itself and the plate; G, 3 in. hole in bottom of launder.



The baffle F banks the slimes and water back in the launder, thereby decreasing the speed of the stream and allowing the sand to fall to the bottom. The sand is slowly washed along until it comes within the influence of the suction of the outgoing stream passing under the plate at D. G is large, so that there is practically no back pressure when once the stream has passed under the plate. H is a launder carrying the sands back to the wheel or tailings pump. If more water is required in H the end of plate at D can be raised by a screw, the movement being taken up by the rubber joint C.

COUNTERWEIGHTED BLIND FOR SANDS COLLECTING VAT.

Description of Sketch II.—The second sketch represents a device to keep the canvas blinds, used in sands collecting tanks, from slipping down, and to keep the blind taut against the slimes gate, thus preventing the escape of sands from the vat; A, is a plate of $\frac{1}{8}$ in. iron, 4 in. diameter, fixed on to the end of a roller by screws and also by a bolt B passing through the centre of the plate and 3 in. into roller, and held in position by a check nut C in a hole checked into the centre of the roller. The nut D is screwed up against C. In eight places around its edge the plate is cut for $\frac{5}{8}$ in. at an angle of 45° to the radius. These cuts are bent so that they form secure catches for the pawl E. E fits loosely on the bolt B, so that it can be moved from one catch to another. E is "set off" so as to clear the nut D, and to come up well within range of the catches. Through the upper end of E is a hole to which is attached by a hook the rope passing over the pulley at the top of the vat down to the counterweight. By means of this the pull of the counterweight comes on the blind and keeps it always free from kinks. The blind is kept from unrolling and coming up by the catches engaging the pawl. To raise the curtain, it is only necessary to place the pawl in the next catch at either end of the roller, when the roller brings itself into position, keeping a continuous strain on the blind.



The President: On your behalf I wish to thank Mr. Dowsett for bringing this matter before us. All these suggestions by practical men are of great value. I understand from the diagram that the sand pulp flows from left to right, so that it gets underneath the narrow opening of the lower plate?

Mr. W R. Dowling: That is correct.

Mr. Jas. E. Thomas: I would like to know if any extra grade is allowed on the main launder. The usual grade for slimes launders is at present 1%.

Mr. W. R. Dowling: I do not know what the grade is. It did not overflow when I saw it.

THE PRECIPITATION OF GOLD FROM SOLUTION BY CARBONACEOUS MATTER.

Mr. W. A. Caldecott (*Past-President*): The fact that gold is precipitated from auriferous cyanide solution by charcoal has been well-known for many years, and finely crushed charcoal has actually been employed to a considerable extent as a precipitant in place of zinc on certain Australian cyanide plants.*

In the ordinary lime kiln, alternate layers of fuel and limestone are fired, so that the burnt lime is usually contaminated with unconsumed carbonaceous matter. Wood is used in some mining districts for burning the lime used in cyaniding, and although such burnt lime usually contains appreciable quantities of unconsumed charcoal, yet the deleterious effects of such a constituent, which increases in proportion to the amount of lime used, do not appear to have been seriously considered.

Many hundreds of tons of burnt lime are used monthly in cyanide plants on these fields, and usually contain unconsumed carbonaceous matter from the coal employed in burning. That the bad effects of this ingredient should be recognised as causing loss of gold, by precipitating it from solution in the sand and slime charges, may be inferred by the following tests among others :—

(*a*) A sample of partly burnt coal from commercial lime was crushed through a 20-mesh screen. 34 gms. of this powdered coal, and 1,105 c.c. of 0·02% working cyanide solution were given 6 hours intermittent agitation in a filled and closed flask, followed by 12 hours contact, when 0·09 dwts. of gold per ton was found in the filtered solution, whilst the carbonaceous precipitate assayed 0·49 dwts. per ton of solution, so that 84·5% of the gold had been precipitated.

(*b*) 17 gms. of coal from burnt lime were crushed through a 30-mesh screen, and added to a litre of 0·021% working cyanide solution, assaying 0·51 dwts. per ton, in a large open dish. The mixture was stirred occasionally for 8 hours, followed by 14 hours quiet contact, when, on assaying the filtrate, it was found that all but a trace of gold in the solution had been precipitated.

(*c*) A glass tube, $\frac{5}{8}$ in. diameter, was filled to a depth of 12 in. with finely crushed half-burnt coal, excluding the −90 mesh grade; from commercial lime, and 2,850 c.c. of working cyanide solution, assaying 1·34 dwts. per ton was slowly percolated through the column during 26·25 hours. The percolate was assayed at intervals and showed only traces of gold.

Tests with fresh unheated coal, showed little or no precipitating effect from a cyanide solution, which appears to indicate that the precipitating agency in coal, which has been partially coked by heat, does not exist in fresh coal, but is probably due to the hydrogen, formed by the decomposition of the hydrocarbons in ordinary coal through distillation, and occluded in the residual solid.

---

* W. H. Gaze's "Practical Cyanide Operations," pp. 121, *et seq.*, 1898.

Assuming that ordinary burnt lime contains only a small fraction of 1% of carbonaceous matter, this is equivalent in a large plant to several pounds of precipitant per day, which comes in contact with dissolved gold in the sand and slime charges, and it is hence obviously desirable to use lime which does not contain carbonaceous matter.

In case it be impossible to obtain lime free from carbonaceous matter, the only remedy appears to be to dissolve the lime required in the mill-service water, before entering the battery. Water can dissolve about 2½ lbs. of calcium oxide per ton, and since only dissolved lime is efficient,* it would thus be possible, though somewhat troublesome, to prevent the carbonaceous matter in the lime from coming into contact with the ore at all, provided that all insoluble matter were settled in the mill service water tanks.

Since carrying out the above experiments, some work has been done at the Knights Deep to utilise the precipitating effect of half-burnt coal, and a good many tons of waste acid solution have been passed through an extractor box containing crushed granular fragments of coal (– 100 mesh + 1,000 mesh product) sorted from the ash dump, with the result that a coal product assaying 12 ozs. per ton was obtained. It has also been found that some base metals, such as copper, lead and mercury are also precipitated from their solutions by coal which has been heated. In view of the results obtained, the application of this method of precipitation has been provisionally protected by patent in the usual manner, and attempts are now being made to develop it further under working conditions.

The President: We must thank Mr. Caldecott for calling our attention to this dangerous element in the presence of carbonaceous matter in lime which is used in cyanide works It shows how careful one ought to be in seeing that the lime used is as free as possible, if not absolutely free, from carbonaceous matter. Referring to one point he suggested, that is the introduction of the lime required into the mill water service before entering the mill. That I do not like. As a millman, I do not want too much lime in the water. It has a deleterious effect upon the plates. There is no question about that. I should prefer to see the lime dissolved in a mixer, and added to the pulp after leaving the mill. Another point has occurred to me. It recalls to mind what I once saw. Some old workings had been partly filled up with ashes. In going back upon these old workings and taking out some of the ore which had been left there, everything was cleaned up very closely, and a lot of this ash found its way in amongst the rock, and on one occasion I saw the battery plates running black pulp for some hours. What the residues of the tank—it was not a very large battery—which was being filled during that time were, I cannot tell you, but I think it must have been a matter to give the cyanide manager a headache.

Mr. E. H. Croghan (*Member of Council*): This brief note of Mr. Caldecott is very interesting, and at present appeals to one a good deal in view of the presence of free carbon in limes. Recently Mr. A. Gillies and myself have been testing various limes, and we find quite appreciable percentages present, varying from 0·1% to 0·2%. We also tried the effect of some of the free carbon separated from local limes, and observed that it is a very definite gold precipitant. In view of this and known chemical facts, the presence of this free carbon is a matter to be considered. Assuming a mine using 1 ton of lime per day; on the above percentages, free carbon to the extent of 2 lb. to 4 lb. per day is introduced. It is obvious, therefore, that in consequence of the proportions thus shown that either carbon free lime (practically impossible to always obtain) should be employed, or that a different system of adding lime should be devised, which will eliminate the carbon factor.

Mr. Jas E. Thomas (*Member of Council*): I should like to point out another objection to using lime in the return water to the mill. It has the effect of choking up the water pipes leading to the mill. The proper way is to eliminate the carbonaceous matter in the lime.

Mr. A. McA. Johnston (*Vice-President*): The figures given by Mr. Croghan bear out results obtained by us, but unless as an indication of the presence of this deleterious substance, the quantitative estimation of carbon present is rather misleading, as part of it may be due to coal, which, as Mr. Caldecott has shown, has no effect on precipitation of the gold from solution. I would suggest that the cyanide manager, or his assistant, can more readily find out whether this charred coal is present by an examination of the lime as delivered.

Mr. H. A. White (*Member of Council*): I should like to know if Mr. McArthur Johnston suggests that the cyanide manager should hand pick the lime.

---

* See this *Journal*, vol. ii., July, 1898, p. 797.

Mr. A. McA. Johnston: For my own part, I should prefer that the cyanide manager take a keener interest in this than the chemist, as it has now been proved that half burnt coal is a precipitant of gold from solutions, and should any pieces of this be found, the consignment ought to be rejected.

Mr. H. A. White: I do not agree with the remarks our President has made, with reference to the deleterious effect of lime in the battery. In fact I had thought this objection, frequently brought forward by millmen in the past, had now become obsolete. When we were on the Nigel mine during this Society's visit, the battery manager, Mr. Charles Manners, pointed out that the alkalinity of their mill water was about ·02%, and their extraction had been improved in consequence. For some time past, on the May Consolidated, we have maintained the mill water at a minimum of ·01% alkalinity, and find the extraction to be improved 2% as compared with a nearly neutral to ·003% alkalinity. We have run up the alkalinity as high as ·06% without any detriment to extraction. I do not assert that the appearance of the mill plates is in any way improved, but that is of no consequence as the increased extraction is a fact borne out both by assays and the actual production of gold.

Mr. Wager Bradford (*Member*): It is a difficult thing to get lime in this country, at the present time, under the present system of burning, entirely free from partially burnt coal, and I am very anxious to know the maximum percentage which may be present in our lime without causing precipitation of gold when it ought not to occur.

Mr. Jas. E. Thomas: I think the answer to that is "none."

Prof. J. A. Wilkinson (*Member of Council*): I am very surprised to learn from what Mr. Caldecott has said that such a very old method of burning limestone, namely, with interposed layers of fuel, still exists in this country. For the students here, who have learnt the newest methods, it will be an unwelcome surprise to find such methods still in vogue. I should, therefore, like to know whether it is possible to have some exact information from Mr. Caldecott as to how the lime is prepared here for use in the reduction works.

Mr. E. H. Croghan: With reference to the cyanide managers looking after the lime and seeing that it is carbon free, I may say that the lime we have taken is absolutely white, and I do not see how it is possible for the cyanide managers to take out this percentage as distinct from lumps of half burnt coal which, at least, have no right to be present.

The President: I should like to set matters right on one point. With regard to what Mr. White has said, perhaps I did not make myself quite clear. I may have given him the impression that I objected to any lime at all in the mill water. That was not my meaning at all. What I meant was, lime in excess. I do not consider ·01 of alkalinity is at all excessive. But if all the lime required in the cyanide works were dissolved in the mill water before passing into the mill, it would not be good for the plates. My experience shows that an excess of lime in the mill water tends to strip amalgam from the plates. As Mr. Thomas also said with regard to choking the pipes with lime deposit, that occurs in some places and does not in others. The same thing may take place with regard to the presence of lime in mill water. Some water is so constituted that an excess of lime does not affect it, whilst other water it does affect as regards the plates.

Mr. W. A. Caldecott: I am glad there has been some little discussion on the points raised in my brief note. I may say that I understand that ordinary limestone has not been burned in this country in retorts, but that magnesite is used extensively elsewhere as a source of carbonic acid. One objection to high alkalinity in mill water which has not been mentioned, which is the danger of settling slimes in the sand collecting vats unless an efficient system of eliminating slimes from the tailings pulp by hydraulic or other classifiers is employed. With regard to Mr. Bradford's point, I may confirm what Mr. Graham has said, that we should endeavour to obtain our lime absolutely free from carbonaceous matter, and that we should urge our friends, the lime burners, to try and supply such lime. I have no doubt that they are making strenuous efforts to overcome this difficulty, and I trust that before long they will be successful. As to Prof. Wilkinson's remarks, I think the solution of the difficulty lies in burning the lime so that the solid fuel itself does not come into contact with the limestone, which should be heated by the gaseous products of the combustion of fuel only. Regarding the effect of high alkalinity in mill water upon amalgamation, I think the views expressed this evening illustrate what we have found in the past, that mill men's experience and opinions upon this matter differ very considerably.

Mr. Wager Bradford: Mr. Caldecott has answered my query, but I am not quite satisfied. I want to know whether Mr. Caldecott has any figures to prove that ·1%, for example, of half burnt coal in the lime used, results in the precipitation of gold in the tanks to any appreciable extent. Is it not possible that you may have a very small percentage of this coal in your lime and still not experience any deleterious influence. Is it certain that the results of the laboratory tests will obtain in a 500 ton tank?

Mr. W. A. Caldecott: Whilst the precipitating effect of carbonaceous matter taken from commercial lime can be readily determined, yet if a lime contains, say, 0·15% of coal this is only equivalent to 3 lbs. per ton of lime, and therefore the amount of coal in the two or three pounds of lime which would be used for a ton of ore, is very small. Hence the amount of gold that this minute amount of coal would precipitate is likewise very small, and a direct determination upon sand or slime is almost impossible, as the limits of experimental error of sampling and assaying would probably be exceeded. Assaying to 0·005 dwt. per ton is beyond the limits of ordinary practice, and yet this trace of gold would amount to a good many thousands of pounds on the tonnage of ore milled yearly on the Rand. To use a parallel illustration, if there were two kinds of cyanide in the market, one of which contained a minute percentage of metallic zinc whilst the other was entirely free from such an ingredient; I do not imagine that there would be much doubt as to which would be generally preferred.

SOUTH AFRICAN COALS AND THEIR ECONOMICS.

By Arthur J. Andrews.

Much has been written concerning the geology of our South African coal beds, both, as regards their age and mode of formation. After pondering over the different suppositions, one might find oneself eliminating as well as piecing together, until some such theory as the following was produced; that there existed, say, in triassic times (the period to which our coal belongs is still a debatable point), enormous tracts of marshy country thickly overgrown by such ultra-luxuriant vegetation such as flourished in the coal ages, and that the products, chiefly in the form of leaves, twigs and light vegetable matter, were deposited or floated down into extensive contiguous lakes, through the alternating agencies of gentle and energetic action.

Applying this theory to our South African coal seams generally, but chiefly considering those in which the Rand is industrially interested, that is to say, local, Middelburg and Natal beds; we may be led to infer that the varying intensity of the transporting conditions, together perhaps, with the variation in the depth of the lakes, must be held responsible for the marked inequality in the value, not only between the relative coal beds, but also as found in the same coal mine.

Presuming then, that the too energetic conditions of transportation prevailing, may have caused the extreme sandiness and inconstant quality of our local coal, it would follow that these conditions were gradually modified in an easterly direction, although nowhere entirely absent; for as we travel, from the Rand through Middelburg, to the eastern coast, we traverse coal fields of progressively improving quality.

The following figures fairly represent the three groups:

	Moisture.	Vol. matter.	Fixed carbon.	Ash percentage.	Evapor.
Local ...	2·04	16·64	63·02	18·30	11·69
Middelburg	1·65	23·03	61·26	13·91	12·64
Natal ...	1·70	25·03	61·09	11·50	13·29

Nearly all South African coal may be classed as bituminous, and can be sub-divided into caking and non-caking. Middelburg and Natal represent the former; local, and pretty well all other South African coals, the latter. The structural arrangement, rather than the chemical composition of the last mentioned coal, probably provides the condition unfavourable to caking, and, in this connection, the large percentage of mineral matter is suggestive. The following notes deal briefly with coal sampling, and certain coal economies. Coal sampling and testing, to be of adequate value, must be effected in such a manner as to prevent loss through the acceptance by consignees, of fuel below the standard agreed upon, and this cannot be accomplished without a well considered system, suitable in its details to the particular requirements of each case.

Fuel tests and analyses are to-day made at the different laboratories on the Rand, with such reliability, that the data produced are accepted as proof, by sellers and buyers alike, of the quality of any coal consignment in question. The facts furnished by the laboratories are generally sufficient, not only to show whether a coal is too high in ash or moisture percentage, and too low in evaporative power, but also to enable a correct estimate to be made as to its suitability for use under the conditions to which it is proposed to subject it. For instance, an analysis showing a high percentage of volatile matter at the expense of the fixed carbon, may indicate the absolute unsuitability of a fuel for use in certain furnaces, and under such conditions of firing as are available. In making use of the figures relating to the evaporative power of a coal as tested in a bomb, it must not be forgotten that a bomb affords ideal conditions of combustion, such as the ordinary furnace at its highest efficiency cannot possibly provide. Nor is it always a simple matter to estimate from the given bomb value of a coal, the relative value it ought to yield in a furnace. Between the evaporative power of a good and equable coal, as burnt in a furnace kept up to its normal state of efficiency; and that given by testing the same coal in a bomb, there will be an approximately constant ratio; but with coals of very irregular quality, more or less badly prepared for market purposes, no true and constant ratio between the bomb and furnace values will exist, and thus the figures furnished by the bomb, cannot be taken as a basis pure and simple, upon which to calculate the evaporation to be expected in the boiler. A coal badly prepared by the colliery is liable to contain a very considerable percentage of carbonaceous shale, varying in degree and quality in different loads and consignments. Were this shale tested under the conditions so favourable to combustion as the bomb affords, it might probably show an evaporation of at least 7 lbs. Such a fuel as this, however, fed to an ordinary furnace, is but semi-combustible, and may even have a minus value, in as much as more heat may probably be absorbed than emitted. Therefore, with such a coal varying in its proportion of included shale, no true ratio between bomb and furnace values can be expected.

The carbon in the shale referred to, is protected from rapid oxygenation by the intimate and enveloping presence of calcite and other minerals, and cannot be successfully attacked unless the shale be first reduced to a powder. Evidence of such half-burnt or unburnt carbonaceous shale can be frequently observed on ash dumps, and has often been taken as the result of bad firing. The following example will serve to illustrate the fact that the bomb value of a coal does not always indicate its furnace value:—

Let A and B represent coal consignments, A being a coal regular in quality and well prepared for the market, and B the reverse of this. The evaporative power of A varies from 11·5 lb. to 11 lb., averaging, say, 11·10, and the boiler efficiency on this coal is 6/10 of its bomb value, viz., 6·66 lb. Now take B as consisting of a coal 4/5 of which is excellent, with an evaporation in the test of 12 lb.; and the remaining 1/5, very poor, with an evaporation of only 7·5. The whole sample tested in the bomb averages 11·10 lb., exactly the same as A, but the 1/5 is too low to be of service in the boiler and must be struck out as useless. Using this coal in the same boiler as before, we now have 6/10 of (4/5 at 12) + (1/5 at 0) producing a final result of 5·76 as the boiler evaporation of B. Thus A and B although giving the same calorific power in the bomb, are in actual practice by no means of equal utility, the one showing a result of 6·66 and the other that of 5·76. It is obvious that the coal-sampler should furnish his laboratory with full and accurate information regarding the condition and appearance of the consignments sampled, as to proportion of shales and fines, as well as supply all details that can be of use. Unless the workers in the laboratory can depend upon accurate and intelligent co-operation on the part of the sampler, the scientific work there performed upon coal samples, with the aim of bringing about and maintaining fuel economies which are everywhere so urgently needed, is in a great measure rendered of no avail.

In sampling coal, the sampler must bear in mind, just as carefully as a gold reef sampler has to do, that a sample to be of any value, must absolutely represent the bulk of the material from which it is taken. Careless and unreliable sampling creates a most incongruous position, for it means that delicate and highly scientific tests and analyses, costly both in time and money, are performed upon samples which are entirely misleading, and are thus worse than useless.

In conjunction with sampling, estimation of quality can be made by inspection by an experienced eye, backed up by a thorough knowledge of the different coals in use; although it must be borne in mind that frequent checking of estimated values by testing is, of course, very necessary. To be able to estimate the value of a coal by inspection, is very serviceable where large quantities of fuel have to be rapidly dealt with.

Estimation and testing of coal may be looked upon as paralleled by the panning and assaying of gold ore. Coal generally arrives at the mines either in short or hopper trucks. To sample the former is a simple enough operation, but it is rather more difficult with the latter. The whole contents of the short truck are easily reached, but not so with the hopper. It frequently happens that most of the fines have been shaken down towards the bottom of the hopper before its arrival at the siding. These fines frequently carry a lower value than the lump coal, whilst it may be, too, that in some cases a poorer coal has been fed to the lower part of the hopper than that which appears at the top. Then again, to take a proper sample as the coal pours from the hopper in emptying, is not often practicable. It is of course quite feasible to take samples from the load, after it has been deposited in the bunker, but a true percentage of fines, as existing in the load, may not be included in the sample. With care and judgment however, to sample a load after it has been dropped into the bunker, is generally the best course to pursue.

A safe and practicable way of obtaining a thoroughly representative sample from a hopper load is as follows:—A good length of the bunker must be clear of coal, the hopper doors are partly opened, and instantaneously the locomotive attached, pushes the hopper slowly over the whole length of the clear bottom. By this expedient a shallow stratum of coal is laid down, presenting an excellent bed for sampling. With coal bought under contract to definite specifications regarding quality, etc., if a system of sampling is not competently and accurately carried out, all the advantage of the contract as arranged is lost to the buyers; and, in fact, the whole matter, from a contract point of view, becomes a farce. Coal bought without any contract or specification as regards the heating value and ash percentage, can be kept up to a proper standard by means of careful inspection, sampling, and testing; moreover, the factor of competition operates in such cases in keeping a colliery up to the mark. It may be neither necessary nor judicious to employ as elaborate a system of sampling and testing in this, as in the former case where coal is bought under contract, but it must be obvious to anyone interested in the subject, that with an absence of any reliable method of investigation of quality of the coal bought, no control of its efficiency can be maintained. In such cases, hundreds of tons of fuel, lower in quality than really arranged for without formal contract, have been sent, and the process has perhaps continued until at last the long-suffering consumee has changed his source of supply by trying another colliery.

In endeavouring to effect economies in coal consumption, it has been the general practice to bring about such by the use of better steaming plants, improved firing, and the purchasing of better, but more costly, coal. Fuel economy, through the proper management of the bunker supplies, has been somewhat neglected. Speaking generally of any commodity, the larger the stock the greater the waste. This particularly applies to a commodity so liable to waste and deterioration as coal is, since to carry a heavier stock of fuel, greater than that absolutely necessary to run a plant, is obviously uneconomical, and constitutes a practice fruitful of. many details of waste, which in the aggregate, involve a far greater financial loss than is probably generally realised.

This brings us to a consideration of the causes of such waste. Primarily, there is the question of the creation and accumulation of fines. Every truckload of coal brings, as a part of it, a certain proportion of fines, and, too often, an undue quantity. This, when deposited in the bunker, is seriously augmented by the comminuting action of the overburden of coal upon that beneath, and by the attrition of coal against coal, as a fresh supply crushes, rolls down upon, and grinds, the underlying stock. When dealing with friable coals, as so many of our South African coals are, the pulverizing action just referred to is most extraordinarily effective. Owing to the weight of its overload, a full bunker suffers, in a modified degree, from these destructive agencies of crushing and rubbing, the former particularly, when the bunker is kept in a state of continual repletion, and the latter unavoidably (but increased to the maximum by a heavy burden of coal) as the withdrawal of coal for firing purposes take place. Besides the making of fines, there is a natural accumulating process going on constantly, whereby the product concentrates in the lower part of the bunker. When the supply of fine coal, manufactured and brought into a mass through the causes touched on above, is at last drawn upon for the furnace, much of it is found to be worse than useless, for when thrown into the fire it absorbs more heat than it gives out, chokes the fire, and falls through the bars unburnt, adding to the quantity of ash and its consequent cost of removal. In connection with the subject of fine coal, it must be borne in mind that it is one thing to have served to a suitable furnace good unoxygenated duff, at the right price for that class of fuel; but quite another matter, to have to deal with a hopeless mass of devitalised carbonaceous dust, which in reality, has been purchased at the price of good round coal. Waste is moreover brought about after the coal is withdrawn from the bunkers (apart from improper firing), by the native fire-boys, who often reduce the lumps unnecessarily, creating at the same time a good deal of fine stuff, part of which, as shown above, falls through the grate unburnt. Mention might here be made of the want of uniformity in the size of coal as generally supplied by the collieries. It is often so large that it must be well broken before going into the furnaces.

Again, there is waste through decomposition. When a bunker is kept full for some considerable time by the frequent dumping of new coal upon old, that part of the coal which is not renewed, decomposes considerably, and the decomposition soon effects the more recent stock. Should there have been a certain percentage of poor shaley coal to begin with (by no means a remarkable circumstance), coal, which at its best, can with difficulty be used, then such weak fuel, after suffering from decomposition, has lost the small calorific value it originally possessed. Decomposition is naturally affecting the coal through slow oxygenation from the time that it is mined, without the assistance of such contributory causes as could be avoided.

It may be interesting here, to make a short quotation on coal decomposition. Kinnear Clark, many years back, wrote the following:—"Coal deteriorates or decays to a greater or less degree by exposure to the atmosphere, by disintegration or crumbling, and also by the gradual combustion of the volatilisable elements. Atmospheric oxygen is absorbed, and converts the hydrocarbons into water and carbonic acid. It has been proved in one case in Germany, that bituminous coal after having been exposed for nine months, lost half its value, as fuel. The decay of coal proceeds more rapidly in hotter climates."

We may therefore, conclude, that the ways in which fuel is wasted are various, and some of them have been referred to in these notes. Faulty firing is a serious one, and doubtless the largest amount of waste incurred thereby, is through the loss of the volatile matter. There appears to be, with of course a good many exceptions, a want of interest shown in firing, so far as the studying of the particular coal in use is concerned: resulting in all coals, whether poor or rich in carbon or volatile matter, being treated exactly on the same lines. It may be that much of this is due to the use, particularly in past years, of very poor fuel, which neither inspires respect nor interest. Another case of bad firing may arise from the employment of native fire-boys, who, very frequently, cannot be properly instructed and interested in their duties, on account of the language difficulty.

All these many causes of waste of fuel: from the mining of the coal, to the evaporation of valuable gases from the smoke stack, may enable electricity to oust steam-power prematurely, from operations which should, from a point of economical production, belong to it for many years to come; and such a revolution would not constitute a true economy, as would be the case should electrical power come into use after a long and keen competition against steam, economically generated.

The President: I have to propose a very hearty vote of thanks to Mr. Andrews for his interesting paper, so well illustrated by lantern slides. I have no doubt he has brought many points before us which will be considered by our members, and I hope there will be a healthy discussion upon them.

Mr. A. McA. Johnston (*Vice-President*): In seconding the vote of thanks and voicing the thanks of our members for the very interesting manner in which he has treated this large and important subject, I should like to draw attention to one or two points not raised by Mr. Andrews, and to confirm his views on others.

The internal wealth of every country is determined by its natural resources; either above, or below, ground; and it seems to me that the duty of every government and of every citizen, is to conserve that wealth for the benefit of the State in the long run, though it may in addition, benefit the individual, in special cases. Now I am well within the mark when I say that there has been, and is now, a great waste in the natural coal resources of the country. Full value in power is not being obtained, and the country's wealth is diminished fruitlessly. One of the results of this wasteful extravagence is seen in the schemes which are being produced to supply the gold mines with electric power, and it is astonishing to note the ease with which the present coal-generated energy on the mines is being replaced. In other countries a harder battle has had to be fought, and there is no doubt but that, were more economic conditions in the use of our coal prevalent here, the installation of electric motive power with its central station, would be longer delayed. Quite recently the United States Government has been engaged in testing the various coal resources of the country, and in determining, practically, the best class of boilers to use for certain coals; with the result that in many cases, hand-fired boilers have been increased in efficiency from 10% to 15% under ordinary commercial conditions. Here, it seems to me, boilers or several types of boilers, are ordered; and then the engineer is expected to gain good results. Little consideration is given to the fact that a coal with, say, 5% ash, 15% volatile matter, and 80% fixed carbon, would burn economically in a furnace which would be quite unsuited for one with 15% ash, 26% volatile matter, and 59% fixed carbon. Or, let us take an actual example: Through some state of affairs (let us call it uncontrollable), an engineer has to change from one coal to another: Typical analyses of two supplies are as follows:—

	No. 1. Per cent.	No. 2. Per cent.
Moisture ...	2·16	6·80
Volatile matter ...	15·12	23·56
Fixed carbon ...	70·36	57·38
Ash	12·36	12·26
(Evaporative factor of coal as received)	12·02 lb.	11·66 lb.

Now, if the boilers were giving economical results with No. 1 coal, most assuredly they were not when No. 2 was used. This then is the first point I wish to emphasise—that the coal wealth of the country can best be utilised by adapting the boiler to suit the coal.

There can be no doubt as to the next point, which is, that to obtain best results, uniformity in the quality of the coal supplied, should be imperative. Note here, two analyses of coal supplied to the same consumer by the same company on successive days.

	Per cent.	Per cent.
Moisture ...	2·20	2·20
Volatile matter ...	26·90	22·70
Fixed carbon ...	52·98	50·26
Ash	17·92	24·84
(Evaporative factor of coal as received)	11 73 lb.	10·28 lb.

Here is seen a difference in evaporative power equal to 1½ lb, and it is to be noted, that the ratio of the value of these coals is not to be expressed as 11·73 is to 10·28, since the poorer coal will give less heat comparatively (as Mr. Andrews has shown), than its better neighbour.

The only way to prevent this variation is to obtain thorough and absolute tests, backed up by penalty contract clauses. The seller thus gains by knowing that he is proportionately paid for good coal, which is above his contract standard, whilst to the purchaser the advantages are, saving on price should an inferior quality be supplied, increased practical interest taken in the supply by the engineer, and also, considerable saving due to, when the quality is uniform; the regular results obtained by regular firing, a special point of interest here since the majority of our firing is done by natives, with not infrequently, careless or inexperienced white firemen in charge. These find the pressure of steam going down during the firing with the poorer quality, and increase the rate of shovelling, but fail to discontinue this when the better stuff comes along. In addition, I would advocate that better monetary inducement be given to trained firemen to continue at this work, and bring it to a high state of efficiency.

As a slight set off against the above analyses I might quote the highest and lowest figures obtained from one coal mine over a period of six months, and over a good number of tests made on this coal.

	Per cent.	Per cent.
Moisture ...	2 60	1·60
Volatile matter ...	27 32	22·96
Fixed carbon ...	60·12	56·82
Ash	9·96	18·62
(Evaporative factor of coal as received)	13·48 lb.	11·92 lb.

The two are slightly extreme, as I think I may say that the variation in this supply lies well within the limits of 1 lb. in evaporative power.

Mr. Andrews has devoted some care and time to the sampling of coal supplies, and the great importance of this, well merits attention. Inaccurate sampling is not only useless, it is misleading, and better had not be attempted. Sampling should be undertaken immediately on delivery, and not after lying in an exposed or heated bunker as the case may be, for some time. It should be as thorough and as representative as are our mine or reduction works samples, and also, the sample should not be stored in the boiler house prior to crushing but should be at once crushed, quartered, and ground, securely bottled, and tested as soon thereafter as possible, so as to eliminate the possibility of deterioration.

You have had pointed out to you the waste occasioned by heavy volumes of smoke discharged from the chimneys. This can also be determined more accurately and with greater control by the resident engineer by means of a self recording CO_2 apparatus. A good average obtained on these fields should be between 12% and 13% of CO_2 registered, though of course, one must remember that this result should not be attained at the expense of driving heat up the chimney. To obviate this, flue charge temperatures should be kept, and comparisons noted, the loss of heat being estimated, whilst at intervals a check on the instrument and determination of CO should be undertaken by the chemist.

In conclusion, I would place before you the analyses of the best South African coal I have tested during the last twelve months. The sample was not picked, but represents the "nuts" from a neighbour colony colliery, supplied to one of our mines.

	Per cent.
Moisture	0·80
Volatile matter ...	20·67
Fixed carbon ...	70 76
Ash	7·57
(Evaporative factor of coal as received) ...	14·53 lb.

The vote of thanks was carried unanimously.

SUGGESTIONS FOR A NEW ATOMIC THEORY.

By James Moir, M.A., D.Sc., F.R.S.S.A.
(Vice-President).

It has long been suspected that most of the substances known as "elements" by the chemist are in reality complex; and nowadays the "atom" of the chemist, is conceived as something possessing a definite *structure*, on which all its properties depend. Speaking for myself, I must say that such a conception is far more satisfying to the mind than the text-book idea of the atom as an indivisible dot with *quasi*-metaphysical attributes.

In this paper I propose to carry this idea further, and to suggest probable structures for the known elementary atoms, by supposing them all to be built up of different arrangements of four or five *primæ materies* (each with definite but entirely supposititious properties); such structural formulæ being selected so as to give the best possible agreement with all the known chemical and physical attributes of the chemical atoms which I am thus submitting to dissection.

Anyone attempting such a scheme as this is naturally driven to found it on the Periodic Law of Mendelejeff—and, as a matter of fact, a great deal of atom-dissection might be done by simply supposing those atoms of higher atomic weight to be made up of the members of the first row in the Periodic Table, with the addition of imaginary structures of constant atomic weight, such as 16, 20, 47 and 88. In this attempt however, I go further than this, and condense even the first row of the table, in order to explain the obvious connexions between the beginning and the end, as exemplified by the following well-known series of compounds

$$\begin{array}{ccccccc} LiH, & CaH_2, & BH_3, & CH_4, & NH_4Cl, & SMe_3Cl, & IPh_2Cl. \\ +1, & +2, & +3, & +4, & +4-1, & +3-1, & +2-1 \end{array}$$

The whole idea of the structures I have invented, is to attempt to explain the selective affinity of the atoms for certain particular other atoms, and to explain this as a matter of shape and fitting together. I find that the first twenty or so can be constructed satisfactorily out of four kinds of *primæ materies* only, and that the rest are repetitions and imitations according to the Periodic Law.

The fundamental conception of my theory is the nature of the carbon atom. We know that all of its four affinities are equal in strength and in space-distribution; therefore, in order to depict carbon, I simply divide the "atom" into four equal sub-atoms of atomic weight three, and arrange these equidistantly in tetrahedral order. If they are assumed to rotate round the empty centre of the "atom," and to attract one another according to the law of gravitation, I believe the system will be stable; in any case, such a system will have four points of maximum external attraction, thus corresponding exactly with the four "bonds" of carbon, and explaining chemical affinity as a mere consequence of gravitation. I propose to call this sub-atom $\left[\frac{C}{4}\right]$ *Zoïkon*, symbol *z*, atomic weight 3·0. Secondly, I retain hydrogen as an element (or sub-atom); its spectrum is simple and inter-related, showing that it is really simple and spherical in structure. I assume in addition however, that its atom possesses an intrinsic repulsive force, so that when it gets attached to a larger structure, the force of gravitation in the latter becomes lessened at the point of attachment of the hydrogen. This is necessary to explain the "saturation" of such a compound as methane (see Fig. I.). This repulsion is probably due to the presence of one of J. J. Thomson's *corpuscles*. Thirdly, I postulate a sub-atom x of atomic weight 2, monovalent like hydrogen, yet not fully saturating another element when joined to it, and therefore capable of shifting its place within the atom-complex. This is necessary to explain variations of valency, and the so-called "residual affinity." In addition, helium and neon are probably true elements, whereas argon, krypton and xenon are not. The atomic weights of the former are still uncertain, and I assume them, for this purpose, to be 4·09 and 19·7 (see below). Finally, I assume the definite metals to contain hydrogen as the cause of their electro-positiveness, and that *per contra*, the subatom x confers electro-negativeness.*

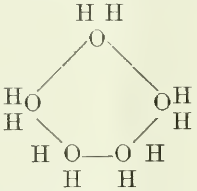
With these simple suppositions one can immediately formulate with some certainty the elements carbon, nitrogen, oxygen and fluorine. Carbon (Fig. V.) has been dealt with above (see Fig. I. for methane, for example). As regards nitrogen, one has to account for (1) its trivalent and pentavalent properties, (2) the spatial equivalence of four of the valencies in pentad nitrogen, and (3) the fact of the combination NR_4 acting like a metal. The formula I propose is an internal condensation of the carbon tetrahedron with the sub-atom x; this latter when inside the tetrahedron leaves its four valencies equal and equally distributed as in carbon, but, in addition, confers an electro-positive property on the whole system, so that when the tetrahedral valencies are satisfied by hydrogen (or alkyl-group), the combination NR_4 or z_4xR_4 acts as a whole and can simulate an alkali-metal (Fig. VII.†). As for trivalent nitrogen, it is only necessary to assume that the sub-atom x is then shifted from the centre, on to one of the points of the tetrahedron (Fig. VI.). The other three points then agree perfectly in situation and otherwise, with the properties of trivalent nitrogen required to explain the oximes, or the similarity of pyridine to benzene, the angle between the valencies being the same as in the trivalent group $:CH-$.

When we come to oxygen, which also requires an increase of atomic weight of 2, we again require two formulæ; one to express its ordinary divalent form, and the other for its tetravalent form (present in the salts of pyrone, the oxonium dyes, etc.). The former is obtained by repeating for a second time the process which gave us the nitrogen formula from carbon, the result being Fig. VIII. and the constitution $O = z_4x_2$. This gives an atom with two main valencies inclined to one another at an angle of 109°; which agrees well with the attributes of oxygen when it occurs as a link in the rings of the heterocyclic compounds. The corresponding tetravalent form with one x in the centre also confirms in a most remarkable manner (see Fig. IX.[*]) the hypothesis recently revived by Prof. Armstrong, that liquid water is $H_{10}O_5$, consisting of five steam molecules (H_2O) joined up into a regular pentagon. The angle of a regular pentagon is 108°, so that there is practically no distortion of the oxygen affinities involved. Water is thus :—

---

* The existence of this element x is already almost predicated to account for certain vacuum-tube phenomena. See J. J. Thomson's "The Corpuscular Theory of Matter," p. 24, middle.

† For convenience of construction in the models the x has been displaced to one side.

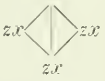


(See also Fig. X.)

in which the tetravalent effect is obtained by simply opening up each of the tetrahedra, and in which the lines joining the five pairs of hydrogens are all perpendicular to the ring.

When we come to fluorine, a slightly different kind of formulation becomes necessary. The atomic weight is five more than that of nitrogen, and eight more than that of boron, and the main valency is one, although the element exhibits marked "residual affinity." I think the simplest supposition is to retain the triangular shape, making it of 3 z with five of the six free bonds satisfied with x's, making $F = z_3x_5$. This accumulation of the anti-metallic sub-atom x will then explain the extraordinary affinity of free fluorine ; and the accumulated "residual affinity" will explain why HF is capable of condensing to H_2F_2 at low temperatures (see Fig. XI.).

The next element is neon ; which, if its atomic weight is really 20·0, need not be assumed to be a true element, since the tetrahedral formula



$$= z_4x_4$$

explains its non-valency quite perfectly. However, to avoid complexity in my scheme, I assume the existence of an element which enters into the composition of the higher atoms and assume this to be neon of atomic weight about 19·7.

I shall next work back from carbon to complete the first row of the Periodic Table. Boron, of atomic weight 11 resembles carbon quite closely, and I formulate it as z_3x, as in Fig. IV. Since its atom is not asymmetric as carbon is, I put the four sub-atoms, of which I suppose it to be composed, in one plane. Note that my formulæ exhibit a rough skeletal analogy between B and F : I think this is justifiable.

The next two elements, glucinum and lithium, are metals and markedly electro-positive ; this property I associate with the presence of hydrogen in the atom ; consequently glucinum is most probably H_2zHe, and lithium H_2zx, possibly arranged as in Figs. III. and II. respectively ; but there are obvious difficulties arising out of such formulæ, such as the fact that there should also be an element of the formula H_2z (divalent, of atomic weight 5), as well as a non-valent substance of the formula zH_4. There may, however, be some reason why elements of such simplicity cannot exist at present, analogous to the non-existence of free methylene, for example. There seems to be no doubt that the atomic weight of glucinum is not 9·0 ; hence I have to assume it to contain a portion of atomic weight 4·1 nearly (helium provisionally).

I have now dealt with the eight parent-elements of the Periodic system, and it remains to indicate what suppositions are necessary to formulate the higher elements. It would be very easy to do this, if one were to assume with Hinrichs and others that all the atomic weights are whole numbers (or multiples of 0·5), but I think it better in the present state of our knowledge, to treat each element separately, retaining the accepted fractional atomic weights. Sodium differs from lithium by exactly 16, and where this occurs I assume the complex z_4x_2 (isomeric with oxygen) to be added ; thus $Na = Liz_4x_2 = H_2z_5x_3$, probably arranged as in Fig. XII. Magnesium has a markedly fractional atomic weight (24·35), and does not resemble glucinum very closely except in divalency. The complex depicted in Fig. XIII. has the required valency, atomic weight (assuming $He = 4{\cdot}09$), and electro-positiveness. If, however, it should turn out that helium has the atomic weight 4·0 exactly, the only other supposition possible is the existence of an element lighter than hydrogen (atomic weight 0·09). This is already suspected in the coronium of the sun's outer atmosphere. It is curious, however, that at this stage in the Periodic table, the atomic weights increase by, roughly, $\frac{4}{3}$, $\frac{8}{3}$, $\frac{4}{3}$, $\frac{8}{3}$; just as earlier in it, the alternate elements increase by 4 (*e.g.*, B – Li, O – C, Na – F, Al – Na, P – Al) ; and I ascribe this to the presence of four helium atoms.

Aluminium, in a similar way to glucinum (which it resembles far more than magnesium does), I assume to contain one helium atom ; the rest of the skeleton given in Fig. XIV., is a duplication of the boron atom (as assumed in

---

[*] For convenience of construction in the models the x has been displaced to one side.

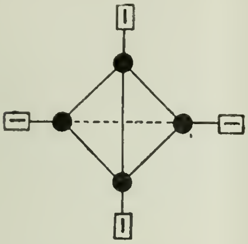


Fig. 1. Methane.

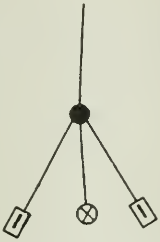


Fig. 2. Li = 7·0.

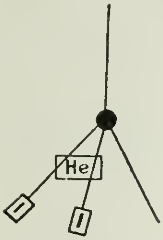


Fig. 3. Gl = 9·09.

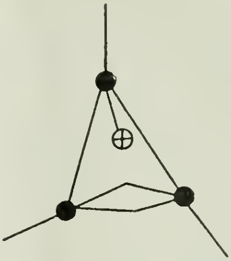


Fig. 4. B = 11·0.

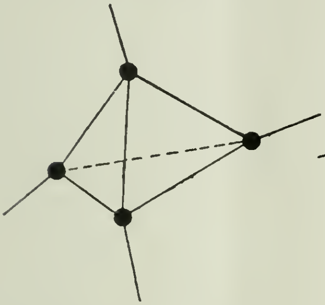


Fig. 5. C = 12·0.

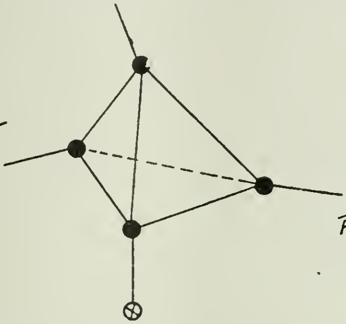


Fig. 6. N = 14·0. (trivalent)

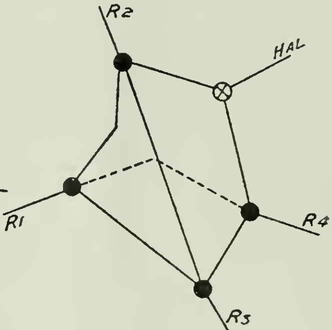


Fig. 7. Pentavalent Nitrogen

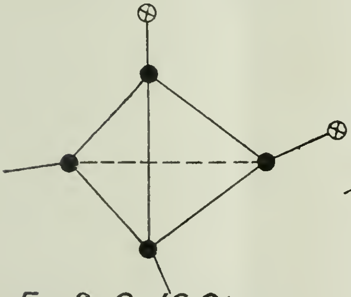


Fig. 8. O = 16·0 (Divalent)

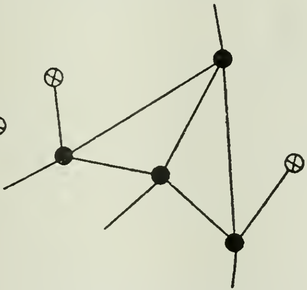


Fig. 9. Tetravalent Oxygen (as in water)

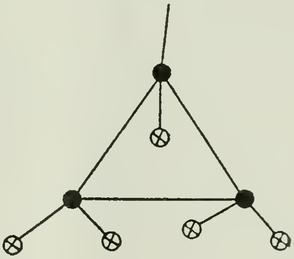


Fig. 11. F = 19·0.

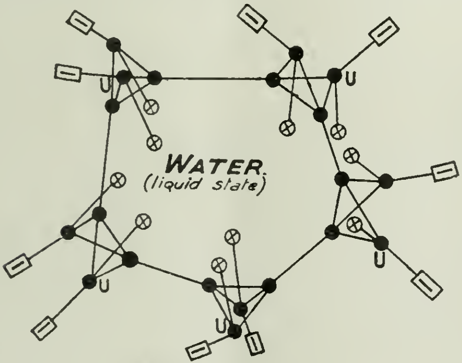


Fig. 10.

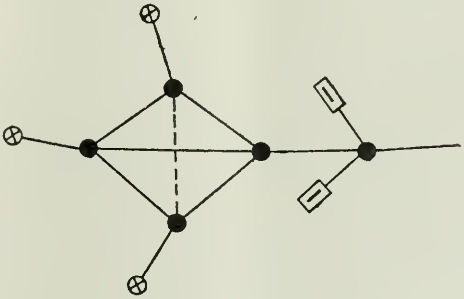
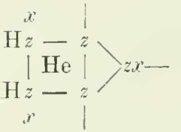


Fig. 12. Na = 23·0

Diagrammatic View of the Models.

Note.—The black circle is z, the crossed circle is x, and the oblong I is hydrogen. In the water model the five dots marked U, with their attachments, are *above* the plane of the paper.

Fig. IV.), modified by the introduction of hydrogen to account for the metallic character of the element. $Al = Hz_6x_2He$. Another possibility not quite so satisfactory is depicted as follows :—



or $H_2z_5x_3He$; this leads to considerable distortion when constructed as a model.

Silicon comes next, and this is almost certainly CHe_4, in which the carbon tetrahedron is distorted by the insertion of a reversed tetrahedron composed of helium atoms (see Fig. XV.); $Si = z_4He_4 = 12 + 16·36 = 28·36$.

The next element, phosphorus, is difficult to represent. It resembles nitrogen in valency, but is almost as little basic as oxygen is. Its atomic difference from nitrogen is the abnormal one of 17·0, and as its atomic weight is a whole number it does not contain helium. The atomic difference can only be z_3x_4 (hydrogen being excluded by hypothesis), so that phosphorus must be z_7x_5, and I suggest the structure given in Fig. XVI. for this element. This is pentavalent with a possibility of trivalency, and also explains the electro-negativeness of the element as due to a certain accumulation of the sub-atom x. Sulphur is a matter of some doubt, because of the uncertainty whether its true atomic weight is 32·00 or 32·08. In the former case it would be Oz_4x_2 (*cf.* sodium), in the latter case (the more likely), it must be as in Fig. XVII., a cube of z sub-atoms with a helium atom inside, and with two of the eight free bonds occupied by x, *i.e*, $S = z_8x_2He$. Chlorine, to account for the fraction in its atomic weight, must contain five helium atoms, and I suggest the symmetrical arrangement given in Fig. XVIII. for its structure (*cf.* fluorine, Fig. XI.). Potassium differs from sodium by 16·1, and therefore must be assumed to contain helium; hence probably $K = Naz_4He = H_2z_9x_3He$, although $H_3z_8x_4He$ (Fig. XIX.) is more easily constructed as a model. Argon may possibly be a true element, but can be also represented as a regular dodecahedron with the sub-atom x at each of its twenty points ($A = x_{20}$), if its atomic weight is really 40 (see Fig. XX.). It is interesting in connexion with argon to recall the properties of the gas SF_6, which is almost equally inert, and has a similar pseudo-spherical molecule.* Calcium is nearly isomeric with argon, but evidently is closely related to potassium. It is probably $H_2z_8x_5He$, or $H_3z_9x_3He$ (Fig. XXI.).

Scandium differs from aluminium by exactly 17, which, as in the case of phosphorus compared with nitrogen, is z_3x_4; hence, $Sc = Hz_9x_6He$. At this stage one notes the interesting coincidence that the addition of 20 roughly to Si, P, S, and Cl gives Ti, V, Cr, and Mn; elements with the same valencies as before, but with a quasi-metallic character. Titanium I assume to be a condensation of silicon and neon, *i.e.*, $Ti = CHe_4Ne$. Vanadium is $Pz_4He_2 = z_{11}x_5He_2$, chromium is $z_{12}x_6He$, which may be looked at both as Sz_4x_4, and as a condensation (with slight modification) of the boron and aluminium atoms [$Cr = AlB_2x_2 - H$]. Manganese, again, has no connexion with chlorine except the isomorphism of $KClO_4$ and $KMnO_4$, and the formula I suggest $H_2z_{11}x_{10} = H_2z_3x_2 \cdot z_4(zx_2)_4$, roughly resembling fluorine, meets the case quite well (it is tetravalent). As for iron, it resembles chromium as well as manganese, and I suggest $H_2z_{12}x_9$ as an approximation, but owing to the difficulty of constructing models of this degree of complexity, it is almost guesswork to decide between the several possible formulæ at this stage. It is a point worth noting here that the eight consecutive elements from titanium to copper exhibit colour in some or other of their compounds; and since the cause of colour in organic compounds is undoubtedly a molecular "squeeze" (conjugated double-linkage or isorrhopy*), a similar kind of distortion must be assumed in the atoms of these eight elements. I assume nickel and cobalt to contain the same nucleus $z_{10}x_4$; in the former combined with H and Ne, and in the latter with Hz_4x_4, the latter structure resembling manganese to some extent.

In the new series commencing with copper we get a compromise between the manganese class and the Li—F series, and the formulæ I suggest are chiefly based on the necessity of preserving these analogies without manipulating the atomic weights of these well-investigated metals. In most cases I assume *small* nuclei with the balance of the atomic weight built chiefly of neon. Copper is really divalent, and I suggest $Hz_5He_2Ne_2$ for its constitution, or possibly $HzNe_3$, so zinc is $Hz_7x_2Ne_2$ or $HzxNe_3$; gallium may be $H_2z_3Ne_3$; germanium is probably C_2Ti; arsenic may be $P(z_3x)_4$; selenium may be $OHeNe_3$ and bromine can be represented as Clz_8He_5 by an addition similar to the one which gave chlorine from fluorine.

We now start on the second "long period" of the system, but the errors in the accepted atomic weights are now considerable, and perhaps give opportunity for more manipulation than is quite justifiable. I therefore state the most probable alternative formulæ for some of these elements.

Krypton, if not elementary, is $HeNe_4$ (82·9); rubidium might be $LiNe_4$ (85·8) (*cf.* NH_4), or $H_2z_4x_4HeNe_3$ (85·2); strontium is $GlNe_4$ (87·9) or $Glz_7x_5He_2Ne_2 = CaHe_2Ne_2$ (87·7); yttrium is

---

* Moissan and Lebeau, *Compt. rend.*, 1900, p. 865; *Journal of the Chemical Society* (London), 1900, abs. ii., 341.

* This is the correct way of spelling "isorropesis," a word coined by E. B. Baly apparently in ignorance of the rudiments of etymology.

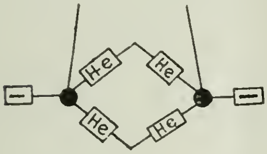


Fig. 13. Mc = 24·35.

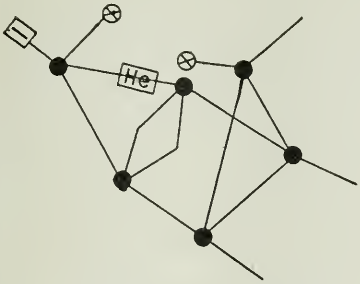


Fig. 14. Al = 27·09.

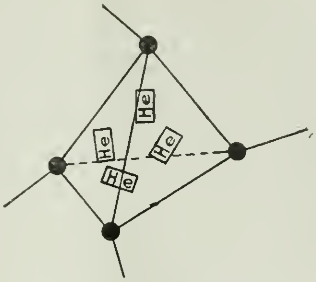


Fig. 15. Si = 28·35.

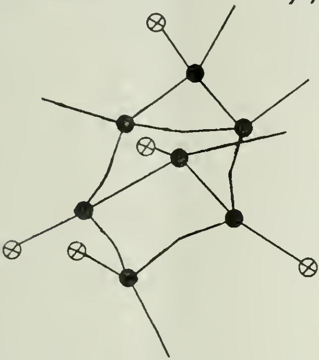


Fig. 16. P = 31·0. (Pentavalent)

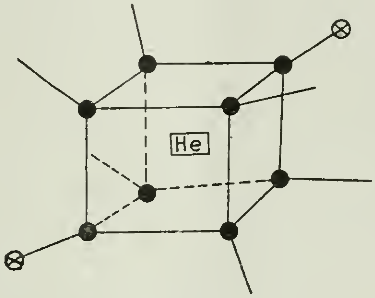


Fig. 17 S = 32·08. (Hexavalent)

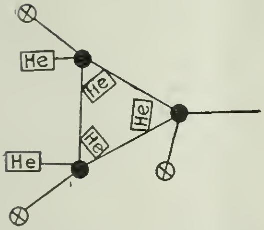


Fig. 18. Cl = 35·45.
(one valency & two others residual)

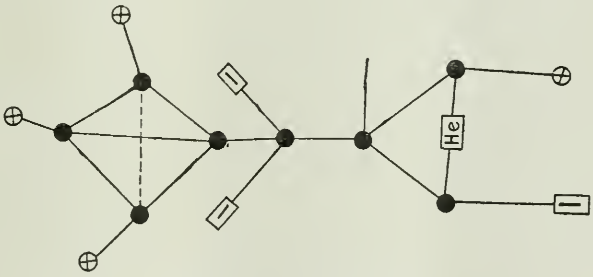


Fig. 19. K = 39·1.

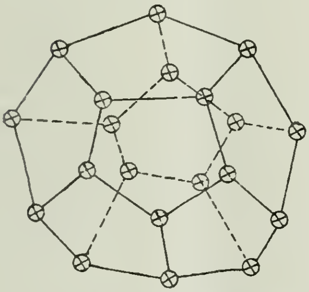


Fig. 20. A = 40 = X20.

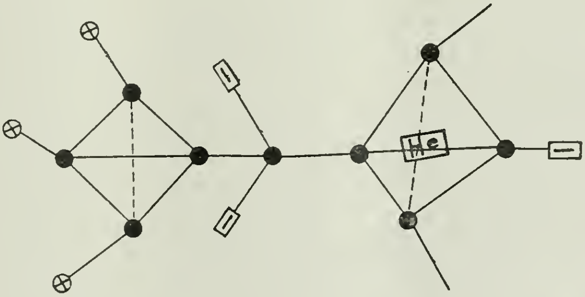


Fig. 21. Ca = 40·1.

Diagrammatic View of the Models.

Note.—The black circle is z, the crossed circle is x, and the oblong 1 is hydrogen.

$\mathrm{GaNe} = \mathrm{H}_2z_3\mathrm{Ne}_4$ (89·8); zirconium is CNe_4 (90·8); columbium is $z_{11}x_5\mathrm{He}_3\mathrm{Ne}_2 = \mathrm{VHeNe}_2$ (94·7) or NHNe_4 (93·8); molybdenum is CrHeNe_2 (95·6) or possibly OHNe_4 (95·8); the unnamed new element of atomic weight 100* from analogy should be $\mathrm{Mn}z_9x_9$; ruthenium is possibly $z_8x_9\mathrm{Ne}_3$ (101·1) (*cf.* Fe); rhodium may be $z_{10}x_7\mathrm{Ne}_3$ (103·2) (*cf.* Ni); palladium, which resembles Co and Ag, is probably $\mathrm{CoHe}_2\mathrm{Ne}_2$ (106·7); silver is $\mathrm{H}z_{11}x_7\mathrm{He}_5\mathrm{Ne}_2$ (107·9) = $\mathrm{Cu}z_8x_4\mathrm{He}_3$; cadmium is possibly $\mathrm{H}z_4\mathrm{He}_5\mathrm{Ne}_4$ (112·3) or $\mathrm{Zn}z_8x_5\mathrm{He}_3$ (111·7) ; indium may be $\mathrm{H}z_7x_3\mathrm{He}_2\mathrm{Ne}_4$ (115) ; tin is probably $z_7\mathrm{Ne}_5$ (119·5) or $z_8\mathrm{He}_4\mathrm{Ne}_4$ (119·2) ; antimony is $\mathrm{As}z_9x_9$ (120) or $z_7x_4\mathrm{He}_3\mathrm{Ne}_4$ (120·1) ; tellurium is $\mathrm{OHe}_3\mathrm{Ne}_5$ (126·7) ; iodine is $z_{11}x_3\mathrm{He}_7\mathrm{Ne}_3 = \mathrm{BrNe}_3 - \mathrm{He}_3$ (126·7) ; and finally, xenon is $\mathrm{He}_3\mathrm{Ne}_6$ (130·4).

Starting on the next "long period," caesium is $\mathrm{RbHe}_2\mathrm{Ne}_2$ (132·8); barium is $\mathrm{Sr}z_7x_6\mathrm{He}_4$ (137·3) $= \mathrm{H}_2z_8x_6\mathrm{He}_5\mathrm{Ne}_4$; lanthanum is $\mathrm{Yt}z_7x_6\mathrm{He}_4$ (139·1) ; cerium is $\mathrm{H}z_4\mathrm{He}_7\mathrm{Ne}_5 = \mathrm{ZrHHe}_7\mathrm{Ne}$ (140·1) ; the next should be $\mathrm{H}z_4x\mathrm{He}_7\mathrm{Ne}_5$, and may be praseodymium or possibly neodymium. I shall omit the rest of the rare-earth elements on account of their insufficient characterisation. Coming to tantalum, I consider it to be $\mathrm{CbHe}_2\mathrm{Ne}_4$ (181·6), whilst tungsten should similarly be $\mathrm{MoHe}_2\mathrm{Ne}_4$ (= 182 9 only, however).

Osmium must have a cubic structure to account for OsO_4, and in this shows a relationship to ruthenium ; it may be $\mathrm{RuHe}_3\mathrm{Ne}_4$ (192·2 ?) ; iridium similarly is $\mathrm{RhHe}_3\mathrm{Ne}_4$ (194·2?) ; platinum and gold are very similar, and are probably $\mathrm{H}z_{11}x_5\mathrm{He}_8\mathrm{Ne}_6$ (194·9) and $\mathrm{H}z_{11}x_6\mathrm{He}_8\mathrm{Ne}_6$ (196·9) respectively (*cf.* Pd and Ag) ; but if the atomic weight of gold is really 197·5, it may be $z_4\mathrm{He}_2\mathrm{Ne}_9$ or $\mathrm{H}z_3x\mathrm{He}_2\mathrm{Ne}_9$; mercury is probably $\mathrm{H}z_4\mathrm{He}_{12}\mathrm{Ne}_7$ (200·0) ; thallium is $z_3\mathrm{He}_9\mathrm{Ne}_8$ (203·4) ; lead $z_7\mathrm{He}_2\mathrm{Ne}_9 = \mathrm{SnHe}_2\mathrm{Ne}_4$ (206·5) ; bismuth is $\mathrm{NHe}_4\mathrm{Ne}_9$ (207·6) ; emanium is probably $\mathrm{He}_5\mathrm{Ne}_{10} = \mathrm{XKrHe}$ (217·4) ; radium is $\mathrm{BaHe}_3\mathrm{Ne}_4 - \mathrm{H}_2 = z_8x_6\mathrm{He}_8\mathrm{Ne}_8$ (226·3) ; thorium is $\mathrm{CHe}_6\mathrm{Ne}_{10}$ (233·5) ; and, finally, uranium is $\mathrm{RaHe}_3 = z_8x_6\mathrm{He}_{11}\mathrm{Ne}_8$ (238·5).

In conclusion, I wish to suggest that my theory, postulating as it does a great similarity between carbon, nitrogen and oxygen, may be able to give better explanations than the current ones of some of the very remarkable isomeric changes known to organic chemistry ; already it explains, for example, fairly well why hydrocyanic, thiocyanic and hydrazoic acids resemble hydrochloric acid, and why NO is possible, and resembles CO (nitrogen as in Fig. VII.). In a similar way the changes of cyanate into carbimide, of carbylamine into nitrile, of ammonium thiocyanate into thiourea, and of benzile into benzilic acid may all possibly be explained by an intra-atomic movement of the sub-atom x, which I have assumed in these formulæ. With these speculations I leave the theory to await the farther extension of knowledge.

REMAINING ELEMENTS TABULATED.

Sc, 44·1	=	$\mathrm{AlP} - \mathrm{N}$	=	$\mathrm{H}z_9x_6\mathrm{He}$
Ti, 48·1	=	SiNe	=	$z_4\mathrm{He}_4\mathrm{Ne}$
V, 51·2	=	PCHe_2	=	$z_{11}x_5\mathrm{He}_2$
Cr, 52·1	=	$\mathrm{SC}x_4$	=	$z_{12}x_6\mathrm{He}$
Mn, 55·0	=	$\mathrm{H}_2\mathrm{FC}_2x_5$	=	$\mathrm{H}_2z_{11}x_{10}$
Fe, 56·0	=	$\mathrm{H}_2(\mathrm{C}x_3)_3$	=	$\mathrm{H}_2z_{12}x_9$
Ni, 58·7	=	$\mathrm{H}z_{10}x_4\mathrm{Ne}$		
Co, 59·0	=	$\mathrm{NiC}x_4 - \mathrm{Ne}$	=	$\mathrm{H}z_{14}x_8$
Cu, 63·6	=	$\mathrm{HC}z\mathrm{He}_2\mathrm{Ne}_2$	=	$\mathrm{H}z_5\mathrm{He}_2\mathrm{Ne}_2$
Zn, 65·4	=	$\mathrm{HCB}x\mathrm{Ne}_2$	=	$\mathrm{H}z_7x_2\mathrm{Ne}_2$
Ga, 70·1	=	$\mathrm{H}_2\mathrm{BNe}_3 - x$	=	$\mathrm{H}_2z_3\mathrm{Ne}_3$
Ge, 72·1	=	$\mathrm{C}_2\mathrm{Ti}$	=	$z_{12}\mathrm{He}_4\mathrm{Ne}$
As, 75	=	PB_4	=	$z_{19}x_9$
Se, 79·2	=	OHeNe_3	=	$z_4x_2\mathrm{HeNe}_3$
Br, 79·9	=	$\mathrm{Cl}_2\mathrm{C}z - x_3$	=	$z_{11}x_3\mathrm{He}_{10}$
Kr, 82·9	=	HeNe_4		
Rb, 85·2	=	$\mathrm{KNe}_3 - \mathrm{CH}$	=	$\mathrm{H}_2z_4x_4\mathrm{HeNe}_3$
Sr, 87·7	=	$\mathrm{CaHe}_2\mathrm{Ne}_2$	=	$\mathrm{H}_2z_8x_5\mathrm{He}_3\mathrm{Ne}_2$
Yt, 89·8	=	GaNe	=	$\mathrm{H}_2z_3\mathrm{Ne}_4$
Zr, 90·8	=	CNe_4	=	$z_4\mathrm{Ne}_4$
Cb, 93·8	=	NHNe_4	=	$\mathrm{H}z_4x\mathrm{Ne}_4$
Mo, 95·8	=	OHNe_4	=	$\mathrm{H}z_4x_2\mathrm{Ne}_4$(?)
? , 100	=	$\mathrm{Mn}z_9x_9$	=	$\mathrm{H}_2z_{20}x_{19}$
Ru, 101·1	=	$\mathrm{FeNe}_3 - \mathrm{CH}_2$	=	$z_8x_9\mathrm{Ne}_3$
Rh, 103·1	=	$\mathrm{NiNe}_2x_3 - \mathrm{H}$	=	$z_{10}x_7\mathrm{Ne}_3$
Pd, 106·6	=	$\mathrm{CoHe}_2\mathrm{Ne}_2$	or	$z_{11}x_5\mathrm{He}_6\mathrm{Ne}_2$
Ag, 107·9	=	$\mathrm{CuO}_2\mathrm{He}_3$	=	$\mathrm{H}z_{13}x_4\mathrm{He}_5\mathrm{Ne}_2$
Cd, 111·7	=	$\mathrm{ZnO}_2x\mathrm{He}_3$	=	$\mathrm{H}z_{15}x_7\mathrm{He}_3\mathrm{Ne}_2$
In, 115·0	=	$\mathrm{GaO}_2\mathrm{He}_3$	=	$\mathrm{H}_2z_{11}x_4\mathrm{He}_3\mathrm{Ne}_3$
Sn, 119·2	=	$\mathrm{GeNe}_3 - \mathrm{C}$	=	$z_8\mathrm{He}_4\mathrm{Ne}_4$
Sb, 120·1	=	$\mathrm{PHe}_3\mathrm{Ne}_4 - x$	=	$z_7x_4\mathrm{He}_3\mathrm{Ne}_4$
Te } 126·7	=	$\mathrm{SeHe}_2\mathrm{Ne}_2$	=	$z_4x_2\mathrm{He}_3\mathrm{Ne}_5$
I }	=	$\mathrm{BrNe}_3 - \mathrm{He}_3$	=	$z_{11}x_3\mathrm{He}_7\mathrm{Ne}_3$
X, 130·4	=	$\mathrm{KrHe}_2\mathrm{Ne}_2$	=	$\mathrm{He}_3\mathrm{Ne}_6$
Cs, 132·8	=	$\mathrm{RbHe}_2\mathrm{Ne}_2$	=	$\mathrm{H}_2z_4x_4\mathrm{He}_3\mathrm{Ne}_5$
Ba, 137·3	=	$\mathrm{SrP}x\mathrm{He}_4$	or	$\mathrm{H}_2z_8x_6\mathrm{He}_5\mathrm{Ne}_4$
La, 139·1	=	$\mathrm{YtP}x\mathrm{He}_4$	or	$\mathrm{H}z_3x\mathrm{He}_7\mathrm{Ne}_5$
Ce, 140·1	=	$\mathrm{HZrHe}_7\mathrm{Ne}$	=	$\mathrm{H}z_4\mathrm{He}_7\mathrm{Ne}_5$
Nd, 142·1	=	$\mathrm{CbHe}_7\mathrm{Ne}$	=	$\mathrm{H}z_4x\mathrm{He}_7\mathrm{Ne}_5$
Ta, 181·6	=	$\mathrm{CbHe}_2\mathrm{Ne}_4$	=	$\mathrm{H}z_4x\mathrm{He}_2\mathrm{Ne}_6$
W, 182·9	=	$\mathrm{MoHe}_2\mathrm{Ne}_2$ (?)		
Os, 192 2	=	$\mathrm{RuHe}_3\mathrm{Ne}_4$	=	$z_8x_9\mathrm{He}_3\mathrm{Ne}_7$
Ir, 194·2	=	$\mathrm{RhHe}_3\mathrm{Ne}_4$ (?)		
Pt, 194·9	=	$\mathrm{PdHHe}_2\mathrm{Ne}_4$	=	$\mathrm{H}z_{11}x_5\mathrm{He}_8\mathrm{Ne}_6$
Au, 196·9	=	$\mathrm{Ag}x\mathrm{He}_2\mathrm{Ne}_4$	or	$z_4\mathrm{He}_2\mathrm{Ne}_9$
Hg, 200·0	=	$\mathrm{CdHHe}_2\mathrm{Ne}_4$	or	$\mathrm{H}z_4\mathrm{He}_{12}\mathrm{Ne}_7$
Tl, 203·4	=	$z_3\mathrm{He}_9\mathrm{Ne}_8$		
Pb, 206 5	=	$\mathrm{SnHe}_2\mathrm{Ne}_4$	or	$z_8\mathrm{He}_6\mathrm{Ne}_8$
Bi, 207·7	=	$\mathrm{bHe}_4\mathrm{Ne}_5 - \mathrm{H}$	=	$z_4x\mathrm{He}_4\mathrm{Ne}_9$
Em, 217·4	=	$\mathrm{He}_5\mathrm{Ne}_{10}$		
Ra, 226·3	=	$\mathrm{BaHe}_3\mathrm{Ne}_4 - \mathrm{H}_2$	=	$z_8x_6\mathrm{He}_8\mathrm{Ne}_8$
Th, 233·5	=	$\mathrm{ZrHe}_6\mathrm{Ne}_6$	=	$z_4\mathrm{He}_6\mathrm{Ne}_{10}$
U, 238 5	=	$z_8x_6\mathrm{He}_{11}\mathrm{Ne}_8$		

---

* *Journal of the Society of Chemical Industry*, 1908, p. 753, and *Chemical News*, 1908, p. 263.

NOTE.—Since writing this, I find I have been slightly anticipated by Messrs. Jessup.[*] These authors also predicate four primitive atoms, viz., H, He and two others known by spectrum lines in the irresolvable nebulæ; the latter may be my sub-atoms x and z; nevertheless, I should point out that my theory will still hold good if the "protons" have the atomic weights ·09, 1, 2 and 19.7, in which case $z = Hx$ and $He = 2x + \cdot 09$.

The President: I have to propose a very hearty vote of thanks to Dr. Moir for his interesting paper, and for the large amount of trouble he has taken in making the models exhibited in order to demonstrate his theory to us, for the extensive original work he must have done in connection with it, and for the lantern slides.

Prof. J. A. Wilkinson (*Member of Council*): I have much pleasure in seconding the vote of thanks. It seems to be quite the fashion among chemists now-a-days to indulge in speculations, which have for their primary object an explanation of the accepted atomic weights as determined by exact experiment. This renaissance of Prout's theory, which flourished about a hundred years ago, has been brought about by the discovery of the Curies and the phenomena which have been brought to light by its means. Before this, however, many relations between the atomic weights had been discovered, and I would especially draw your attention to one, which appeared in an abstract in the *Journal* in August, 1908. I must claim your indulgence here for a brief repetition of this uncommonly curious fact, which is almost haunting in its weirdness. If the elements are arranged in the order of increasing atomic weights and then numbered consecutively, by multiplying the even numbers by two, and the odd numbers by two and to this adding one, we get numbers equal to the atomic weights, the maximum error in only two cases being equal to one. The first element, hydrogen, is not counted in the series. For example:—

				At. Wts.	Error.
1.	*Hydrogen*			*1*	
2.	Helium	4		4	
3.	Lithium		7	7	
4.	Beryllium	8		9	− 1
5.	Boron		11	11	
6.	Carbon	12		12	
7.	Nitrogen		15	14	+ 1
8.	Oxygen	16		16	
9.	Fluorine		19	19	

This rule extends as far as iron, beyond which its discoverer states that it fails in its application (*vide* abstract). The explanation of such a fact and the many other relations which have been discovered awaits a Darwin who will do for chemistry what Darwin himself accomplished for the biological sciences. Recently, in a paper read before the London Chemical Society, a very short abstract of which appears in the Proceedings (vol. xxv., No. 351, p. 26), the atomic weights of the first fifteen elements have been calculated according to a simple formula, which with a slight modification gives those of the next thirteen elements, the agreement being in nearly all cases to the second place of decimals. No further details are given except a short discussion by the President, Sir William Ramsay, who stated that in his opinion this was an epoch making paper, and we must, therefore, await the publication with great interest and expectation. It is impossible at the moment to criticise the views the author has put forward, based on the subdivision of the carbon atom and the hypothetical element of atomic weight 2, but I hope to have the opportunity later. For a moment, however, our attention has been diverted from metallurgical and mining problems to the theoretical foundations of the science which underlies them, and our thanks are due to the author not only for his lucid verbal exposition of his hypothesis but also for the trouble he has taken in making and explaining the illustrative models before us.

Mr. F. W. Watson, B.Sc., F.I.C. (*Member*): Our thanks are due to Dr. Moir for bringing this important subject of the atomic theory before us in a new light. When one considers what strides were made in chemical knowledge when Dalton's atomic theory was accepted, and used as a good hypothesis, any fresh ideas on the subject are always most welcome if they by any chance throw light on properties of certain atoms about which we are at present in comparative darkness: this is especially so with valency.

Change in valency of an element has always been a problem of great difficulty to the theorist in chemistry and, as far as I am aware, there has been no satisfactory hypothesis put forward so far to explain this phenomenon. Theories have been advanced as to the form or shape of an atom, such as the vortex theory of Lord Kelvin, but these theories have not attempted to give atoms a constitutional formula as used in our conceptions of molecules of chemical compounds, nor have they disputed the fact that the elements we know at present are real elementary substances and not complex.

In the face of this the author's diagrams of the atoms of what we suppose to be elements, are somewhat staggering, and require careful study before we can quite realise their significance. From some of the diagrams it would appear that nitrogen and oxygen are to be considered as derivatives of carbon, direct additive products, if the diagrams are used in the same manner as ordinary graphic formulæ. This certainly is an idea that will cause, if accepted, some consternation and much study to most teachers of chemistry: and many a course of lectures, which, with slight modification, has done duty for session after session will have to be revised and rewritten.

---

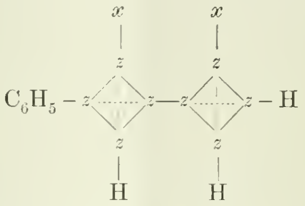
[*] *Phil. Mag.*, 1908, 21, and *Annual Reports on the Progress of Chemistry*, vol. v., p. 1.

As might be expected, the author bases his hypothesis on the Periodic Law. When Mendelejeff first drew up this table he anticipated, as has been subsequently proved in some instances, that the number of elements would be increased and the gaps filled up. Now the author's idea is to reduce the number of elements to five, and the first thing one notes is the choice of these elements. About hydrogen and carbon, nothing need be said. They are well known substances, and nothing has come to light to prove that they are not elements. Why, however, should substances such as helium and neon be retained as elements in preference to all others? Comparatively little is known of these substances and, as the author states, the atomic weight even is uncertain. I do not think that the author has given in his paper sufficient reason for the choice, the only reason seemingly being the inactivity of these substances Then we come to the postulate made in the earlier part of the paper, namely, the existence of the unknown quantity x and the unique property attributed to it by the founder of the theory, namely, that it is monovalent and yet is not capable of completely saturating another monad. This is, to the best of my knowledge, an entirely new conception of an atom or sub-atom, if the latter term may be allowable. The supposition of the existence of the unknown x is *the* point in the author's theory which is most open to debate and discussion.

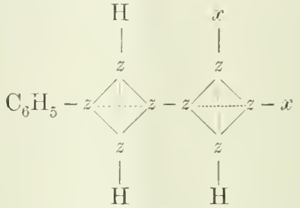
I would like to call attention to the proposed structure of the nitrogen and oxygen atoms; they are carbon atoms with the addition of, in the first case, one x sub-atom and, in the second case, two x sub-atoms. Both, therefore, would be derivatives of carbon, and it will also be noted that the ratio between the atomic weights of hydrogen and oxygen would be put down as exactly 1 : 16. This ratio has been the subject of some most exact experimental work, and has not been found to be exactly 1 : 16; resulting in two tables of atomic weights being used at present, the one with the base H = 1 and the other with the base O = 16. The author's hypothesis would make these two tables one, which would be most convenient, but hardly justified by facts.

If nitrogen and oxygen are to be regarded as additive products of the carbon tetrahedron some strange cases of isomerism will be observed. Take the two bodies, phenyl-hydrazine, $C_6H_5 \cdot NH \cdot NH_2$ and benzyl-alcohol, $C_6H_5CH_2OH$.

Writing these formulæ according to the proposed notation, we have in the first case

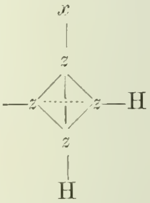


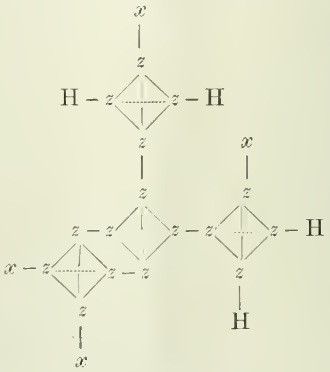
The second gives :—



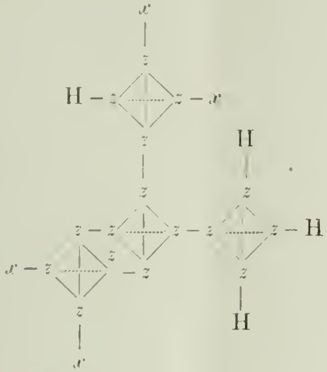
(I have written the phenyl-radicle as C_6H_5 in each case instead of extending it in tetrahedra.)

These bodies would be isomeric, and this is improbable when one considers the remarkable property of phenyl-hydrazine of forming hydrazones with aldehydes and ketones. In this formula of phenyl-hydrazine we also notice a carbon tetrahedron with one valency linked to x, one to C_6H_5, one to H and one to : thus giving us an asymmetric carbon tetrahedron. According to van 't Hoff's hypothesis, therefore, there ought to be a dextro-rotatory and a levo-rotatory modification of phenyl-hydrazine : these optically active isomers have not at present been discovered. A most remarkable example is the case of urea and acetic acid. Writing these according to the proposed notation, we have :—





Urea, $NH_2 \cdot CO \cdot NH_2$.



Acetic acid, $CH_3 \cdot CO \cdot OH$

These would be isomeric, and the noteworthy point is that, to our certain knowledge, one is a base and the other a strong acid. In the author's paper it is stated that the addition of a hydrogen atom confers an electro-positive tendency, and the addition of x confers an electro-negative tendency, so it would seem curious that in two bodies with such similar constitutions, an exactly opposite reaction should be found.

The vote of thanks was carried by acclamation.

THE THEORY OF BLASTING WITH HIGH EXPLOSIVES.

(*Read at October Meeting, 1908.*)

By E. M. Weston, A.S.M.B. (Member).

REPLY TO DISCUSSION.

I have very little to add to my paper, and to the valuable contributions brought forward by several members. There are, however, several points requiring further emphasis. Mr. Cullen's paper is a splendid example of the kind of research I pointed out the need for, and his experiments carried somewhat further will help to solve many of the queries I raised. Might I suggest that some experiments be made, placing the detonator in the middle of the charge, and in the bottom. Also the plan I suggested for use with charges of interposed tamping alternate with the explosive, might be experimented with first, in long charges of six or eight plugs. I suggested that detonators, one or more, without any fuse, should be placed in, say, the 3rd and 6th cartridge, counting down the hole. My theory was, and as a theory I suppose it is as good as any other in our present state of ignorance, that the detonation of each detonator by the on-coming wave of detonation through the blasting gelatine, would reinforce this wave, and thus carry a more perfect detonation to the bottom of the charge. I certainly hold with Mr. J. Whitehouse that the alleged beneficial effect of using clay instead of sand tamping as regards more perfect detonation and greater efficiency is not proved, however great an economic improvement it may be to adopt Mr. Leslie's plan, and make all tamping cartridges of clay or slime with proper appliances on the surface.

Mr. C. D. Leslie's contribution on the practical economics of blasting requires little comment from me. I am, however, glad to see that the gospel of long holes I have persistently preached since I came to this field, finds approval from such high authorities. Mr. Leslie's suggestion to use a copper end to tamping sticks, and to remove the end paper on cartridges before charging is a hint worth adopting.

Mr. H. M. Thomas has given us a splendid thesis on my statement, that the art of blasting consists of noticing these heads and planes of weakness and of taking advantage of them. In my contribution to Mr. Tom Johnson's paper* I pointed out that the rock here does not break in the crater form, "beloved of theorists," but tends to tear off in slabs from foot to hanging wall. Mr. Thomas has evidently made a close study of the occurrence of these heads, and I have no criticism to pass on his most valuable and practical paper, except to note that I am surprised to learn that it ever was the custom to drill back holes 6 in. longer than front holes. The practice is contrary to every law of blasting and to common sense. Evidently Mr. Thomas has proved that in the Jumpers Deep these heads occur with sufficient regularity to base regular work upon them. Personally, I should have been inclined to think that in most mines, owing to rolls, dykes, and minor and major faults, this regularity was not so marked. Mr. C. D. Leslie evidently also considers that it remains to be proved that these cleavage planes are everywhere so well marked and regular. In Mr. Thomas' remarks on stoping, I am also gratified to note that it is now conceded that it is folly to keep machines cutting under pillars, and in tight toes of stopes, and that, as I stated in a paper four years ago†, hand hammer work should be used to help machines to work more efficiently.

Mr. H. C. F. Bell is right in believing that water is not really a good tamping. I would, however, warn him seriously if he values his professional reputation, not to plead guilty to the practice of putting interposed tamping in his holes. It has been most solemnly condemned on "a priori" grounds by experts too numerous and eminent to mention, and even if it does break extra rock—it should not, —and if the last bottom cartridge does not remain unexploded, it has been laid down that it ought to do so. The same warning applies to Mr. J. Moyle-Phillips and Mr. Tom Johnson. Mr. Phillips, I think, is mistaken regarding the relative rate of detonation of dynamite and blasting gelatine. The base of both these explosives is the same, *i.e.*, nitro-glycerine, hence the beneficial effect he inferred that he obtained was probably due to the fact that in the ground in which he was working, dynamite was more suitable than the more powerful blasting gelatine.

---

* See this *Journal*, vol. viii., May, 1908, pp. 349-352.
† See this *Journal*, vol. vi., Aug., 1905, p 42.

I must thank Mr. Tom Johnson for his serious discussion to my paper, which is in itself a gentle rebuke for my somewhat flippant contribution to his "Notes on Rand Mining." I am glad that he agrees with me in my criticisms of some of Messrs. Daws' theses. I note his correction. 12,000 kilos per sq. cm., should, of course, be about 159,000 lb. per sq. in.

Regarding the loading of blast holes, quite an animated discussion has been going on in the *Engineering and Mining Journal* on this subject. Some believe in placing the primer in the centre, some in the bottom, and some on the top of the charge. The various advantages and disadvantages of all three methods are well set out.

Mr. G. C. McFarlane, whom I have before quoted as an authority on blasting, has a most interesting article, giving the latest American practice. I give some extracts without any abridgment.

"As everyone knows, dynamite is detonated from a solid into a gas by pressure. When the detonating cap lets go, it compresses about 2 cub. in. of the surrounding mass of dynamite sufficiently to cause dissociation ; the transition from solid to gas must occupy a definite interval of time, about one-twenty thousandth of a second. The initial explosion of one or two cubic inches of dynamite in the immediate vicinity of the cap would generate sufficient force to compress the entire charge to the dissociating pressure."

"With dynamite, detonation is accomplished with a succession of jolts, separated by imperceptible but definite intervals of time. This explains why a strong cap and a well packed charge give definitely better results. The strong cap detonates enough of the surrounding charge to make a strong primary explosion. When the charge is well tamped, the primary explosion would compress 2 or 3 ft. of powder to the pressure of dissociation. With a weak cap and poorly compacted dynamite, it is conceivable that it might take three or four jolts to explode the whole charge."

"The reason the ground around the primer is badly shattered is that the primary explosion of an ounce or so of dynamite has not power enough to spring the rock burdened on the hole, and as the tamping is tightly up against the explosive, the pressure crushes and shatters. In the interval that precedes the detonation of the main portion of the charge the gas from the primary explosion blows the tamping back an inch or so and cushions the main explosion to a certain extent. For these reasons I believe a cap should be in the top of a charge, because it is evident that the main or secondary explosion tears the rock out. With a cap on top, a small part of the upper end of the charge is rendered useless for rock breaking purposes ; with the cap in the bottom a small part of the bottom of the charge is consumed in shattering the bottom of the hole. With a lifter or back hole it would not make a particle of difference, as far as breaking ground went, where the cap was ; with a cut hole, where the power is needed on the toe of the hole, a cap on top of the charge will tend to put the locus of the principal concussion a fraction of an inch nearer the bottom of the hole."

"In breaking rock with mud-caps, I find it makes a very material difference whether or not the detonating cap is placed on top of the dynamite. The most effective way to break a rock is to pick out a little depression and form a wall around it with plastic clay ; fill this crater with dynamite, lay the cap on the centre and cover the whole with soft clay. If the cap is shoved down into the dynamite the breaking power of the mud cap is lessened. The only possible explanation of this phenomenon is that when the detonator is near the bottom of the dynamite, the detonation lifts a portion of the explosive bodily from the rock before it bursts into gas. High explosives must have a good contact with rock which it is desired to break. When a mud cap is placed on top of a stone, the explosive is usually plastered down tight against it, and the stone is broken into several fragments ; if the same quantity of explosive were poked under the stone it would have poor contact and do very little execution. This is the reason why dynamite is supposed to strike down."

"In regard to tamping a hole, I think that ordinarily 8 or 10 in. of clay tamping is enough. Strong cut holes require more tamping to get the full power of the explosive. Take a case of a 6 ft. incut hole in the face of a drift, the hole being 1¾ in. in diameter and loaded with 32 in. of 60% explosive. The centre of the charge would be burdened 4 ft. and in hard granite or trap it would take one two-thousandth of a second for the concussion to travel from the explosion to the free face. This means that for a period of one two-thousandth of a second the gases of the explosion must be confined. At the end of the period, the momentum imparted to the rock tears it away from the solid. During this period the tamping will be subjected to a pressure of 450,000 lb., sufficient to force 18 in. of clay or 3 ft. of water tamping 6 in. This shows that 18 in. of clay tamping is none too much. A 6-ft. hole burdened 30 in. would kick 8 in. of clay tamping back 6 in. before the rock cleaved from the solid."

"I think the best results are obtained by putting in 18 in. of tamping for the incut holes and 14 in. for the other holes. Using a 2 in. time interval on the fuses, the fire in the fuse will be several inches in the tamping before an adjacent hole goes off. It is no uncommon thing for a hole to kick off 2 or 3 ft. of another hole; as long as the fuse was burned down into the tamping and firmly bedded in a foot or so of clay, the spent fuse would break in case a collar blew off; otherwise the fuse might be pulled out of the hole."

"If the ground is only moderately hard it pays to drill the cut holes so as to bottom $1\frac{3}{4}$ in.; then 4 lb. of 60% or 75% dynamite can be put in the bottom 30 in. of each hole. If the ground is extremely hard, time can be made by drilling the holes smaller, bottoming them $1\frac{1}{4}$ or $1\frac{3}{8}$ in. Not only do small steels cut faster and hold their gauge better, but it is also far easier to collar a hole on a slanting face with a small starter."

"When small holes are used it will be necessary to spring the four cut holes with about four $1\frac{1}{4}$ in. sticks of 60%, then blow them out well with a blowpipe; this will chamber the holes so that they will take enough powder to throw out the incut."

"If a hole that is solidly tamped with about a foot or so of clay tamping misses fire, it is certainly no joke to dig the tamping out. If the caps are always soaped, and waterproof fuse is used, such mishaps are bound to be extremely rare. One method of drawing tamping that is a little safer than picking it out with a scraper, is to burn it out by repeatedly loading up whatever collar there is to the hole; after two or three shots the concussion will jar through the tamping and set off the main charge. Another method, which I believe is safer, is to make a little tank by capping both ends of a 6 or 8 in. pipe about 3 ft. long, tapping in a couple of inch nipples on opposite sides of the pipe. This tank would hold two or three pails of water. To blow tamping use a small tip on the blowpipe, wedge a plank or board across the drift, and after inserting the blowpipe in the hole, lash it fast to the board so that it cannot kick back. Now run the blowpipe hose to the water tank and run the machine hose from the air pipe to the upper nipple of the tank. Use a valve 200 ft. or 300 ft. back from the face for turning on the air pressure. This method works well in an upper or flat hole; it will often wash out a foot of tamping without shifting the blowpipe."

Some of these statements are quite new to me, as that the time of detonation is one two-thousandth of a second, and that detonation proceeds in a series of "jolts," so that the detonation of the primer can be treated separately to that of the main charge. If these theories are sound, it is obvious that in certain holes the more tamping we can use and the greater its cohesion the better the result; but in other cases with long holes where we require the force of the escaping gases (according to my theory) to do some rupturing work towards the collar of the hole, a lighter tamping is an advantage. Note also the statement that the wave of concussion preceding blasting takes one two-thousandth of a second to travel 4 ft. in granite and similar rock, and that for this period the tamping must resist the full pressure of explosion over its inner area.

Generally, this discussion has emphasised my statement, that there is much room for well thought out experiments with a view to giving an answer to the questions raised, and to improve our general practice.

A FEW NOTES ON CUPELLING GOLD-LEAD BULLION.

(*Read at November Meeting, 1909.*)

By Geo. Melvill (Member).

REPLY TO DISCUSSION.

As my notes brought forth very little discussion I have only a few remarks to make.

Mr. Dowling mentions using a little flux, after he has blown off the litharge, to collect the remaining portions of litharge. This plan is quite sound, and the base metal would be still further reduced if he cupelled a few bars of clean lead before adding the flux.

In reply to Mr. Rusden, I use an ordinary Sirocco fan. When I use the shallow channel I fill up the deep channel and hole with bone ash (wet), and so far have never had a leak. My tests before being used are practically dry, as they are as a rule, nine months old, so there is no danger in putting in the lead when the fire is lit.

Since writing my notes I have had another record test. It was dried very slowly for nine months, and lasted six months, during which time I did 18 cupellations, and cupelled 440,900 oz. bullion (=412 bars), an average of 23 bars per cupellation.

No. 1	hole did	5	cupellations and	94	bars
,, 2	,,	5	,,	121	,,
,, 3	,,	7	,,	177	,,
,, 4	,,	1	,,	20	,,
		18		412	

After the 18th cupellation the test had a crack in the breast so I did not like to risk any further cupellations. My coal consumption, including the lighting of fire, is about six bags for 21 bars. When a new test is used a little more coal is necessary for annealing.

GRADING ASSAYS AND GRINDING EFFICIENCIES.

(Read at December Meeting, 1908.)

By Arthur Yates, A.I.M.M. (Corresponding Member of Council.)

DISCUSSION.

Mr. H. Stadler *(Associate)*: By chance, the paper by Mr. A. Yates, of Sumatra, read before this Society, and a paper of mine, read before the S.A. Association of Engineers, re-opened in the same month the discussion on the most important question of computation of crushing efficiency, about two years after this question had first been approached by Messrs. S. H. Pearce and W. A. Caldecott.

Since Mr. Yates (as I also do) considers discussion, or even criticism of a paper, as an attention paid to it and to the Society, no apologies will have to be made if I venture to make a few remarks as far as its mechanical part is concerned.

Like Messrs. Pearce and Caldecott, Mr. Yates bases his efficiency calculations on Rittinger's theory, by which the relative work done in crushing is claimed to be proportional to the surfaces exposed in crushing. As for given weights of ore, the total surface area of all the particles in each lot varies inversely with their average diameter, the surfaces exposed by each grade and, therefore, the values for the relative work done, are obtained by dividing the weight by the average diameter of particle in that grade. This so-called "law" is merely the application of a simple mathematical deduction from the taking to pieces of a cube to smaller cubes. Any student of elementary mathematics knows that the total area (A) of the smaller cubes produced from a unit is (if $S =$ sides of the cubes) $= \frac{6}{S}$, or for a given weight (W) $\frac{6W}{S}$. Rittinger's formula gives therefore simply the relative values for the absolute values obtained for cubes. It is certainly attempting to take the exposure of surfaces as a criterion for the work done, and this is also correct as long as, under the work done, the output or the increase of the commercial value of the product is understood, but it would be utterly wrong to mistake it for the mechanical work done.

The area of fractures being one-half of the area of surfaces, the relative values are for both, the same, and they represent therefore at the same time, the area over which the cohesion of the molecules has to be loosened, or the "Power" required, to cause the fracture of the body. The second factor determining the mechanical work, the "distance" the power has to run through, is evidently a fractional co-efficient of the diameter of the particle, but as these diameters decrease in the same proportion as the surfaces increase, *the product of "power by distance" or the mechanical work, is a constant for each of the grades of a crushing scale based on a geometrical curve.* The same conclusion is also arrived at by the application of the law of Prof. Kirk, stating that the energy required to cause the fracture of a body is proportional to the volume.

The relative values for the mechanical work done are therefore readily obtained by multiplying the weights on the different grades with the figures of any arithmetical progression with which the grades are to be numbered.

Although hardly an objection can be raised against the conclusive force of these arguments, a discussion may be admitted on the correctness of the *way* to introduce the distance into the calculations, but in any case it will never be permissible to accept a method of calculation as correct, in which such an important factor is cheerfully disregarded and neglected. Indeed, the area of surfaces or fractures exposed, increases with the fineness of the grades to an enormous extent, and if the mechanical work done were really proportional to the surfaces exposed, practically all the work done in crushing would be represented in the finest grades only. The author, himself, is well aware of the unreliability of the figures obtained from the surfaces in the finer grades, and he gives an excellent example in his paper of how the more or less arbitrary assumption of the average diameter of the smallest particles in the -200 grade, brings about great differences in the efficiency figures. In cases like tube milling, where the range of crushing extends over a few grades only, the error made by taking the surfaces alone as a criterion for the work done, is hardly observable, but the consequences of irrationally sinning against an eternal physical law become quite apparent, if a larger range of grades is to be considered, or if fine and coarse grinding products are to be compared. Taking for instance, the pieces of banket ore fed to the tube mills, and assuming that their reduction be done in two stages; a first stage from their original size of, say, 4 in. average diameter to the size of tailings as fed to the tube mill, and a second stage from this size to the finished product as represented by the discharge pulp. Calculated on the basis of surfaces, the relative work done would be 112 for the first stage and 655 for the second stage, whilst the correct figures obtained by the rectified method of calculation are 24·6 in. and 5·5 in. respectively. There cannot be the least doubt that the first figures, by which the hard work done in reducing the 4 in. pebbles to sands of the size of tailings, is credited with only about one-sixth of the work done in further reducing these tailings by a few grades, must be utterly wrong, and that the figures obtained by the rectified method, showing almost the reverse, are more likely to be correct and conform to practical experience.

The inaccuracy of our instruments of measurements of the finer grades, and the enormous influence of the arbitrary assuming an average size of the smallest particles of the – 200 grade, would probably have proved to be an insurmountable obstacle in the computation of crushing efficiency, but by the very fortunate circumstance that in the rectified method the two factors of the product "*power by distance*" are exactly inversely proportional, all these uncertainties become at once eliminated, and the anxiety at seeing our calculations upset by some cigarette ashes fallen into the – 200 grade, or by a draught causing a little more dusting, has completely lost its terror. The factor of accuracy being practically the same for all grades, the grading analysis can fearlessly be done more summarily, if only a sufficient number of well assorted screens be taken. Consequently, for the purpose of mechanical investigation, such great accuracy in screening, as practised by the author on his mine, would be of no use, and even an error of a few percentages will not affect the result, especially if the displacement is only from one grade to the next adjacent one. In presence of a larger amount of slimes, however, the case is different, and greater care must be used, as the slimes adhere to the coarser particles and the displacement of the weight on the grades takes place from the very finest to the coarser grades. This circumstance is the principal reason for the marked difference of the grading by wet and dry screening, as shown by the author: but as long as the metallurgists are satisfied that no further classification be made in the – 200 grade between very fine sands and slimes, this drawback can easily be overcome by washing out not only the slimes, but also the finest sands up to the vicinity of the 200 screen. For the Rand ore there is a clearly marked point, where this limit can safely be drawn, without danger of interference with the + 200 grade. By this mixed method the discrepancy between the wet and dry method is reduced to a minimum, and in addition to greater accuracy, the screening is done much more quickly, in spite of the necessity of double drying, for which all mines are well equipped. In presence of a certain amount of slimes, the operation of screening, especially in the finer grades, never comes to an end, whilst with washed sands, in comparatively a short time, a point is reached where nothing passes any longer through the screen.

In these times, where, encouraged by the now generally recognised success of the tube mills, a still further increase of the capacity of stamps by stage crushing is seriously ventilated, be it by the interpolation of other grinding machines, such as Chilian mills, etc., or by tandem tube milling, and in presence of the imminent advent of bought engine power, whereby any improvement in energy consumption means immediate cash saving, the question of computation of crushing efficiency has become a most important phase of metallurgical work. Much remains to be done; even now, after four years of practical experience in tube milling, our leading millmen still disagree on all material points as to the best system of working to be adopted for obtaining the best possible return for the energy spent. Also in regard to stamps, we are still awaiting from one of the advocates of heavy stamps some information as to how far this policy is based on technical considerations, and how far the increased output keeps pace with the costs for increased energy consumption.

ROUTINE ASSAYING ON A WESTRALIAN MINE.

(*Read at December Meeting, 1908.*)

By W. B. Blyth (Associate).

DISCUSSION.

Mr. L. J. Wilmoth *(Member)*: The author is to be congratulated on his interesting paper, and it is to be hoped that members will come forward and discuss the various points raised.

It would seem that Westralia has made but little alteration of late in the method of preparing her mine samples for assay. I can remember identically the same method the writer describes being used on a well-known Kalgoorlie mine some six or seven years ago.

Like Mr. McArthur Johnston, I would not care to place much reliance on the checking by the duplicate samples mentioned. All samples, such as mentioned, should be finely ground before any duplicate be taken. For mine samples a 60 should be used at the coarsest.

The author mentions that he reduces his reduction work samples to pass a 60 linear mesh.

This, I believe, is common practice in Kalgoorlie, but is it fine enough? The usual Rand practice is to reduce the samples still finer, and I think if the writer refers to some of the back numbers of our *Journal* he will see the matter has been thrashed out by some of our most able members.

The size of the muffle used seems small for the work done, and must have a tendency to retard the speed of the work. As a rule, a larger muffle can be used than the furnace is designed to take, and I think the author would be well repaid if he tried this alteration.

The stock flux mentioned, I think could bear slight alteration. The author mentions the dusting when ordinary Battersea crucibles are used. This is due to the use of an excess of soda in the flux. With such a small amount of litharge in the flux, after the fusion the slags must be almost lead free. Personally, when assaying similar material I prefer to have my slags carrying a considerable amount of lead, as they pour freer and cleaner. Some little time ago I used the following flux on a perfectly clean quartz—not banket—with excellent results.

Ore, 1·0 A.T. ; soda carbonate, 1·0 A.T. ; litharge, 2·0 A T. ; borax, 0·5 A.T.

The use of borax I agree with, and in reply to Mr. White's remarks will—with your permission—give some comparative figures of the two methods, "borax" and "non-borax," in the fluxing of an ordinary Rand banket. These comparisons were made at the time when Mr. Hogenraad's paper on "Experiments in Fire Assaying in Sumatra,"* was under discussion.

First, 24 charges were weighed from a sample which had been most carefully mixed by rolling. These were then fluxed as follows :—

(1) Six with 1½ A.T. soda and 1½ A.T. litharge.
(2) ,, 3 A T. of a flux consisting of soda 1, litharge 1, borax ⅓.
(3) ,, 2 A.T. soda and 2 A.T. litharge.
(4) ,, 4 A.T. of the same flux as in No. 2.

In all cases 1·0 A.T. of ore was taken.

Lots 1 and 2 were placed in one fire, and lots 3 and 4 in another. In the first fire the lot 2 fused in 25 minutes, but of lot 1 the first five formed a scum like mass in the crucible, and were altogether useless. The sixth crucible poured, but was extremely pasty and most unsatisfactory. This is the non-borax flux, so strongly advocated for banket samples by Mr. H. A. White. All the contents of the crucibles of lot 2 were fused, but on pouring were inclined to be thick In the second fire, lot 4 fused down rapidly, whilst the lot 3 showed more or less action for a further 15 minutes after the fusion of lot 4. On pouring, both were satisfactory, lot 3 was inclined to be thick, though slightly more fluid than lot 2. Lot 4 poured like water. The buttons were detached and cupelled, the average results being as follows :—

Average	Lot 1.	Lot 2.	Lot 3.	Lot 4.
	40·2 dwt.	43·3 dwt.	45·8 dwt.	47·9 dwt.

In the case of lot 1 the figure given is the result of one assay—the other five being useless. From the above figures it would seem, that the bulk of the flux as well as its nature, exerts some influence on the accuracy of the results. The non-borax flux in the proportion of 3 of flux to 1 of ore was found unworkable, but by increasing the proportion to 4 to 1 the charge poured with ease. On a second glance it would seem that the more fluid the slag the higher the assay results. To confirm results between lots 3 and 4, ten samples were weighed up from another lot of ore, five of which (lot A) were fluxed the same as lot 3, and five the same as lot 4. The average results were as follows :—

Lot A.	Lot B.
44·6 dwt.	45·9 dwt.

It would seem from these few assays that the use of borax in the flux is desirable, as its presence has a tendency to yield high results, due it would seem by the observations during experiment, to the extra fluidity it lends to the charge.

In regard to the method used for the assay of cyanide solutions, the quantity of solution taken is so small that I fail to see how an accurate estimation could be made of the gold contents in low grade samples, such as from bottoms of zinc boxes. The method described is rapid, but I would prefer evaporation with litharge, or the copper sulphate method, using in either case 20 A.T of solution at the least.

The assay ton system, using shillings as a basis is excellent and has many advantages, avoiding as it does the subsequent conversion of pennyweights into monetary values.

The method of bullion assaying differs considerably from that used on these fields. The amount of sample operated upon being so very small as to raise the question of a printer's error in printing "grains" instead of "grammes."

In using magnesia cupels for bullion assaying, I would take the liberty of drawing the author's attention to the wide range of gold absorption shown by even the same make of cupel. A very important factor in this is the variation of temperature. I have known two cupels, one not 2 in. in front of the other, show as much as 25% difference in absorption. Certain brands of magnesia cupels show this difference due to temperature considerably more than others, so care is needed in selecting the particular brand to be used.

---

* See this *Journal*, vol. viii., Sept., 1907, pp. 73-77 ; Nov., 1907, pp 150-151.

The Secretary read the following correction which had been received from the author.

THE DISTRIBUTION OF THE GOLD PRODUCED ON THE RAND.

By T. Kirke Rose, D.Sc., A.R.S.M., M.I.M.M., F.C.S. (Hon. Member).

Dr. T. K. Rose: In dealing with the imports of gold bullion into the United Kingdom the gold contained in imported ore was not taken into account. The value of this was £4,137,334 for the five years, 1903-7. The gold is extracted and converted into bullion in England, and must be counted as part of the available supplies. There are other items not accounted for which need not be specified, but the net result is that I now estimate the total amount of bullion (apart from coin, old material, imported gold-leaf, imported watch cases, etc.) used in the industrial arts in the United Kingdom in the years 1903-1907 as £8,400,000 or £1,680,000 per annum. Thus about 5·4% of the Rand gold was used in the industries in these years, and 30% for coinage, instead of 4·2% and 31·3% as given in the paper.

THE PILGRIM'S REST GOLD FIELDS AND MINING METHODS.

(Read at March Meeting, 1909.)

By J. Moyle-Phillips (Member).

DISCUSSION.

Mr. A. Richardson *(Member of Council)*: The author's general sketch gives a faithful representation of the district in the vicinity of the mines of Pilgrim's Rest, and illustrates what the difficulties of transport are. To those accustomed to the undulating monotonies of the Rand, who wish to see what the Transvaal can produce in the way of scenery, a visit to the edge of the Drakensberg can be recommended, particularly to that portion stretching from Belvedere northwards; here the views are all magnificent and some of them are superb. Pilgrim's Rest may be regarded as the parent of gold mining in the Transvaal, the first gold field in the country having been discovered there. This discovery was made by Mauch and Baines, who subsequently, in 1871, claimed the reward offered by the Government for the finding of payable gold. This incident deserves a passing comment, as it shows there was some enlightenment in official circles even in the '70's. Unfortunately the sovereigns of the concessionaire proved highly acceptable in the then necessitous state of the Transvaal exchequer, with the result that "the Lydenburg district was practically locked up for fourteen years owing to the concession policy."* That the potentialities of this gold field are not so well known to day as they ought to be, must in some measure be due to this prolonged period of slumber. It is pleasant to know, however, that some of the early diggers, before the introduction of this exclusive policy, were able to lift fortunes out of the creeks, 200 oz. nuggets being not all uncommon.

Regarding the lithological characters and mode of origin of the reefs, I cannot do better than quote the conclusions of A. L. Hall, whose admirable monograph on the geology of the district is contained in the Report of the Geological Survey for the year 1906:—

"The reefs of the dolomite series in the neighbourhood of Pilgrim's Rest belong to the class of interbedded quartz reefs—that is to say, they follow particular horizons in the general sequence of beds, and hence might equally well be termed ore sheets. All the breaks, undulations, or structural irregularities generally, which have affected the dolomitic strata, have been equally shared by the auriferous zones. The most essential and invariable constituent of the reef is whitish yellow to dirty rusty brown quartz, always opaque. Pyrites is another fairly common mineral, but the better class of ore is practically free from this sulphide, and indeed the more dirty brown and earthy the reef becomes, the more valuable it turns out to be. Where the reef becomes locally strongly pyritic, or contains much copper pyrites, as in portions of the Clewer and Peach Tree Creek mines, the gold contents may rise to 1 oz. or more per ton, though this advantage appears to be lost by the difficulty in treating such ore in the cyanide works. Taking the whole area into consideration, the reefs yield from 9 to 13 dwt. per ton, though in certain places this is easily exceeded. Thus in the Clewer mine, near some of the breaks caused by dykes, the ore is distinctly richer.

Regarding the mode of origin of the gold, certain interesting deductions seem justifiable. In the first place, it is clear that the reefs originated after the deposition and consolidation of the associated sedimentary strata, for we know of no process which could cause the contemporaneous deposition of crystalline silica having all the appearance of vein quartz. It is possible that mineralised solutions, carrying silica in the soluble form of colloidal silicic acid, perhaps derived from the chert layers in the Dolomite, dissolved some of the latter with simultaneous deposition of vein quartz in such a way that a slow interchange of material took place, for one cannot suppose that the reef now seen is due to the infilling of a system of fissures in nearly horizontal strata. In what form the gold was conveyed, it is impossible to say, for neither visible gold, nor the telluride of this metal, are known from the reefs. In any case, the constant presence of iron-pyrites is not without significance, considering the ease with which gold is precipitated from its solutions by ferrous salts. It has been shown, experimentally, that pyrites particles are capable of precipitating the precious metal from its solution. Now iron is one of the most common base metals, carried in underground solutions, and below the permanent level of ground water, where there is a deficiency of oxygen, such salts would easily be reduced to the ferrous condition, and hence cause the deposition of gold particles. The subsequent decomposition of the pyrites is easily accounted for by the fact that successive stages of denudation have removed portions of the ore sheets out of the belt of cementation, into that of weathering, above the permanent ground-water level. Here the underground waters are charged with oxygen and carbon-dioxide, which remove the sulphur of the pyrites and some of the iron in the form of soluble sulphates, carbonates, etc. Hence the present zone of oxidised reef must sooner or later merge into an unoxidised zone where pyritic ore is constant, this would not necessarily affect the value of the reefs, but would necessitate a modification in the treatment of the ore after it leaves the mine."

---

* J. P. FitzPatrick, "The Transvaal From Within" (popular edition, 1900, p. 46).

The same writer is also of opinion that the reef below Bevets reef, at Frankfort, is the Beta reef, and that systematic search may disclose the true Theta reef, as there are indications of another auriferous horizon in the dolomite ; he also goes on to say that there can be no doubt that the Theta and Beta reefs belong to two distinct auriferous horizons in the upper and lower Dolomite respectively

There is a mode of occurence of gold which the author has briefly referred to, and that is the "digger's leaders." These consist of stringers of quartz cutting through the strata at intervals of about 50 ft., and varying in thickness from a thread up to 6 in. They are barren in the diabase, poor in the dolomite, carry about 2 oz. to the ton in sandstone, and from 10 to 40 oz to the ton in the slate, but this richness appears to occur only in the neighbourhood of a dyke. I have seen some beautiful specimens of leaf gold taken from these leaders, some recalling the Gilead P. Beck of romance and his famous golden butterfly.

It would be interesting to know how far from the outcrop the oxidised zone extends and, as a matter of curiosity, to what distance in, the furthest workings have penetrated, also whether the difficulty of drilling in dolomite is due to the toughness of the rock preventing the cutting edge of the drill from chipping it away as readily as it does in our harder but more brittle quartzite. The particular method employed for resueing also, I think, merits a full description, for unless great care is exercised there is always apt to be a loss of fines amongst the packed waste, and in a case like this, where the earthy matter in the reef is the gold carrier, the fines must be very fine and very valuable.

The prices quoted for mining poles are higher than ours on the Rand, these for poles, 7 ft long, being approximately:—8 in., 3s. 8d.; 7 in., 3s. 2d.; 6 in., 1s. 9d.; 5 in., 1s.; and 4 in., 8d.; and in view of the cost of timbering a preliminary treatment, at least for the timber used in the main tramming roads, might in the long run more than pay for itself in the lower costs for labour and renewals.

Regarding small propositions, the author does not mention the fact that alluvial has been worked for several years at Ross Hill and is still being worked there ; a digger is charged nothing for the right to work, but all gold must be sold through the company. From what I can hear about it there is nothing in it for either party, and the saying "once a digger always a digger" may account for its continuance.

The scarcity of water, concerning which the author quotes such alarming figures regarding the Blyde, prohibits any washing on a large scale.

The terms mentioned for leases seem to be the usual ones ruling everywhere, and in Rhodesia 10% of gross winnings is, for a five stamp equipment, usually taken as being equivalent to 25% of the profits.

One company operating in the Lydenburg district allows the prospector who finds payable reef to peg out about 50 claims for himself, reserving the option to buy him out for about £10,000, plus the reasonable cost of effective development done by him, should he elect to sell. In connection with the running of small plants the suction gas engine is looked upon very favourably, and some more particulars regarding it would be acceptable, especially as to first cost, cost of erection and maintenance.

The author is to be congratulated on having given us such an instructive paper, and I hope some other member in the district may be induced to write one on the reduction practice, as the treatment of the coppery ores of Pilgrim's Rest must present many points of interest.

The President: Before closing the meeting I should like to announce that the Secretary has received an invitation for the Society to visit the Simmer Deep property on the 22nd of May next. I am sure this is a treat we have been looking forward to for a long time, and I hope a large number of members will avail themselves of it.

NOTES ON PRECIPITATION.

(*Read at March Meeting, 1909.*)

By MATHER SMITH (Member).

DISCUSSION.

Mr. J. Hayward Johnson (*Member*): The author is to be congratulated upon opening up a subject that has been lying dormant for some considerable time, and his paper resolves itself more into a matter of, time the solution is in contact with the zinc, than, the depth or amount of Zn required to a ton of solution.

The old style boxes were generally built with compartments of about 30 in. to 36 in. × 18 in. to 22 in. × 24 in., and in the old times of sands treatment this was sufficient, when it was customary to pump or gravitate solution on to the sands without any draining dry until the time was required to drain for discharge, the solution was allowed to leach from the tanks just at sufficient speed as not to crowd the precipitation boxes, (one case I know of where solutions gravitated on to the tanks and drained through the boxes at such an even pace that no attention was needed for days), I believe if we referred back to those times we should find that the solutions were in contact with the Zn for periods of 20 to 30 minutes, very likely more; this appears to be the period the author's solutions are in contact with his zincs, whereas in many cases now, even with boxes having compartments 5 ft. × 2 ft. × 2 ft. 6 in., the solution is only in contact for a period of 6 to 8 minutes, and I think that on some plants where the long narrow boxes are still in use, we should find the time of contact much shorter. I think it is a pity this subject has not come up for discussion before, as it might have saved many boxes being erected where length was aimed at instead of width. Depth and amount of Zn shaving used per ton of solution has to be increased when time of contact decreases. Regarding area and depth of Zn required to solution, I believe it is possible to obtain excellent precipitation with only an inch depth of shavings as long as sufficient time of contact is allowed. At the same time the author's suggestion that all precipitation boxes should be built in the same proportion as those in use by him, would necessitate a box receiving 200 to 250 tons solution per 24 hours, to be about 12 ft. square. I believe I am right in stating that one reason for building the old long narrow boxes was to allow for a good filtration of the solution before leaving the boxes to prevent the possibility of fine particles of gold bearing zinc being carried into the sumps.

On plants where slimes are not treated, to restrict the compartments of the precipitation boxes to five would be rather drastic, my experience has proved that the first and sometimes second, third and fourth compartments have acted as cleansers of the solution for the remaining compartments, especially in the box receiving the first solution from a tank which has been filled with sands that have been crushed with (very) acid water, and also a strong likelihood of some native compound refuse water as well.

It is a pity the author has not stated the strength in KCN of the solutions entering his boxes, it would be interesting to know, also as a comparison to the strengths of solutions entering the boxes in the old days, previously mentioned, when the strong solution entered the box at ·2% KCN to ·3%, medium solution at ·13% to ·2%, and weak solution at anything below ·13%. At this time precipitation was not the trouble but extraction.

The meeting then closed.

Proceedings

AT

Special General Meeting, April 17, 1909.

Prior to the Ordinary General Meeting a Special General Meeting of the members was held to consider a proposed alteration of the rule regarding student members.

The members present were the same as at the Ordinary Meeting.

The President: You all know the purpose for which this meeting is called, and I will ask Mr. Thomas, if he will be kind enough, to give us his views and reasons for this alteration of rule.

Mr. Jas. E. Thomas (*Member of Council*): In the past there has been no particular qualification required of any person on becoming a member of this Society. The soundness of this policy is very evident, as we have members all over the world who take an active interest by contributing papers to be read before the Society, and to the discussions. These discussions have always been open to contributions from persons not members of the Society, and I am sure in the future, as in the past, we will always welcome discussion by any visitor qualified to speak on the subject before the meeting.

Now it seems to me that, as the different mining groups have decided on employing a large number of white workers, we should extend the benefits of the Society to those workers, and as the majority are not in a position to afford the subscription to become Associates, we should admit them on the "Student" basis. It is pretty certain that such Students will, later on, become Associates and Full Members of the Society, as their pay increases.

The objection has been raised that some might remain "Students" after they could well afford to become Associates, but that is obviated by their having to make fresh application each year on the form "C" in which the amount of their salary is certified by the head of the department in which they are employed.

Students not only benefit by the discussion of papers but also by the abstracts taken from all the best chemical, metallurgical and mining papers in the world and published in the Society's *Journal*. These abstracts, I am sure, we have all found most useful in the past.

It is obvious, I think, that the adoption of the alteration of this rule will not only benefit those who take advantage of it to become "Student Members," but will also prove of benefit to the Society, so I have much pleasure in proposing that the amended Rule 9, as follows, be adopted :

Rule 9 as now in force :

9.—Students.—Any *bona fide* student of a recognised technical school, college, or institution of University standing shall be eligible to become a Student upon making application in the form provided in annexure "C" hereto, and being duly vouched for by his professor or teacher. Students shall be admitted by the Council. Student Membership ceases, *ipso facto*, upon the completion of the student's course of study at his technical school. Students may become Members of the Society by being proposed, approved, and elected in the manner provided for in Clause 7. Students shall have all the privileges of Members, with the exception of holding office and voting.

Rule 9 amended as proposed :

9.—Students.—(*a*) Any *bona fide* student of a recognised technical school, college or similar institution, or (*b*) any man engaged as a *bona fide* apprentice or learner in mining or metallurgical operations, shall be eligible to be admitted a student member upon making application in the form provided for in annexure "C" hereto, and being duly vouched for by his instructor or by the head of the department in which he is employed. Students shall be admitted by the Council. Student membership ceases, *ipso facto*, under (*a*) on the completion of the course of studies; and under (*b*) at the end of each financial year, but it may be renewed on making a fresh application on the form "C." Students may be admitted as Associates or elected Members of the Society by complying with the provisions of Rules 7 and 8. Students shall have all the privileges of Associates.

Mr. K. L. Graham (*Member of Council*) seconded the resolution.

The President : I think the alteration will meet with our approval generally. We certainly ought to try and encourage the younger workers on these fields who have been taken on in the battery and cyanide works, and do everything we can to help to train them up, and get them interested in their work. I would like to ask if any member has any amendment to bring forward or wishes to speak to the motion.

There being no response, the President continuing said : Apparently it meets with your approval, and I will therefore proceed to put it to the vote. Before doing so, I will call your attention to Rule 58, which provides that alterations or additions to the rules shall only be made by a majority of two-thirds of the votes of the members present at a Special General Meeting convened for the purpose. Therefore I must request every member to vote either for or against.

Mr. A. Whitby (*Member of Council*) : Has the matter been referred to our legal advisors ?

The President : No, I do not see there is really anything that needs to be submitted to our legal advisors.

The motion was then put to the vote and carried unanimously.

The President : I may say that personally I am pleased this rule has been carried, and I think it will not do any harm to the Society, but should tend to increase its sphere of usefulness.

The Special General Meeting then closed.

The Annnal Dinner.

The annual dinner of the Society was held at the Carlton Hotel, on Saturday night, April 24th, the President, Mr. R. G. Bevington, presiding. There were also present :—

Messrs. G. Albu, F. F. Alexander, G. R. Airth, W. Beaver, S. H. Boright, Wager Bradford, A. D. Buckeridge, J. W. Buckley, T. J. Butcher, E. W. Buxton, W. A. Caldecott (Past-President), W. Calder, W. Carr, Palmer Carter, Chas. Chudleigh (Mayor of Johannesburg), F. W. Cindel, J. Coats, J. Cowie (Secretary, Chamber of Mines), J. P. Creed, E. H. Croghan, M. Cullen, J. S. Curtis, Hon. W. Dalrymple, M.L.C., W. H. Dawe (Vice-President, Chamber of Mines), Prof. J. H. Dobson, J. B K. Dodds, W. R. Dowling, P. L. Edwards, Lewis Evans (President, Association of Mine Managers), J. F. Ferguson, M. Ferguson, J. S. Fisher, Sir Percy FitzPatrick, M.L.A., A. S. Gillies, E. Goffe (Vice-President, Transvaal Institute of Mechanical Engineers), G. Goodwin, W. S. Gordon, K. L. Graham, J. Gray, J. C. Greer, Jr., D. W. Greig, R. E. Hall, H. Hamel, J. Hanna, G. Hauton, H. F. Hayllar, W. Beachy Head, Dr. J. McC. Henderson (President, Geological Society of S.A.), A. Heymann, C. H. Hilditch, C. B. Hilliard, W. G. Holford, E. Hollis, W. L. Honnold, J. H. Hopkins, Sir W. van Hulsteyn, M.L.A., J. l'Ons, E. H. Johnson (Past-President), A. McA. Johnston (Vice-President), G. Kent (President, Rand Pioneers), T. Kerr, C. B. Kingston, H. G. Kirkland, J. W. Kirkland, Dr. M. M. Klein, B. Kopelowitz, R. N. Kotzé (Government Mining Engineer), E. J. Laschinger (President, S.A. Association of Engineers), J. B. Latto, G. A. Lawson, Hy. Lea, Jas. Lea, Q. J. Leitch, A. Lichtenstein, T. L. Lockhart, Dr. D. Macaulay, M.L.A., A. G. Maclatchy, J. McCracken, Jas. McPhee, J. N. Meeser, Hans Meyer, H. Mitchell, J. T. Mitchell, W. E. C. Mitchell, Dr. J. Moir (Vice-President), P. T. Morrisby, Dr. H. Temple-Mursell (Vice-President, Transvaal Medical Society), J. Munro, S. Newton, W. H. Odgers, D. F. Ogilvie, F. B. Ogle, C. F. Parry, E. H. Parry, S. H. Pearce (Past-President), Dr. G. Perkins, O. P. Powell, Sir Thomas Price, K.C.M.G. (General Manager C.S.A. Railways), L. Pryce, A. Purnell (President, Pharmaceutical Society), J. F. Pyles, R. Raine, *Rand Daily Mail*, L. Reyersbach, A. Richardson, A. M. Robeson, G. C. Robinson, H. Fraser Roche, N. Rogers, A. Rosendorff, H. Rosendorff, Fred. Rowland (Secretary), Major C. B. Saner, J. Sexton, G. O. Smart, Prof. G. H. Stanley, *Star*, A. W. Stockett, W. H. Stout, J. R. Streeter, S. Sykes, F. Taylor, H. Taylor, J. A. Taylor, R. M. Taylor, W. Taylor, A. Thomas, J. E. Thomas, W. E. Thorpe, M. Torrente, *Transvaal Leader*, J. A. Vaughan (Chief Inspector of Machinery), J. F. Walker, A. Wallace, F. W. Watson, A. Whitby, H. A. White, J. Whitehouse, Prof. J. A. Wilkinson, L. J. Wilmoth, E. J. Wiseman, Prof. J. Yates (Past-President), and J. A. Yule.

The President gave the toasts of "The King" and of "His Excellency the High Commissioner," both of which were enthusiastically received. He then announced that letters regretting inability to attend had been received from General Botha, General Smuts, Sir George Farrar, Sir Julius Wernher, the Hon. J. de Villiers, the Hon. J. Rissik, the Hon. H. C. Hull, Mr. Lionel Phillips, Mr. Otto Beit, Mr. J. W. S. Langerman, Mr. J. G. Hamilton, Mr. F. D. P. Chaplin, Mr. W. F. Lance, Mr. B. Kitzinger, Mr. W. Adye, also a telegram from their Past-President, Mr. W. Cullen, at present in Capetown, and from the President of the Transvaal Institute of Mechanical Engineers, Prof. J. Orr, who, he regretted, had had to go to Caledon for his health.

Mr. R. N. Kotze (Government Mining Engineer) in proposing the toast of "The Chemical, Metallurgical and Mining Society of South Africa," said: "I must express my deep sense of the honour done me. When I look round on this distinguished gathering and at the past achievements of this Society, I feel that this toast might have been entrusted to abler hands and a more eloquent tongue than mine. Of course, gentlemen, the record of this Society is a noble and distinguished one, and when one looks back at the past history of these fields, such, for example, as is recorded in the recent paper by one of the members of the Society, Mr. Coombe, entitled "Reminiscences of the Early Days of the Rand," one feels the vast progress which has been made; and one feels that a great deal of this progress has been due to the efforts of members of this Society. Well may you be proud of its record. Its history and achievements are recorded in the volumes of your *Journal*, which, I think, properly forms a treasured portion of the library of every one of its members. In looking over these *Journals* one can see the varied activities of the Society in the past, and in looking over the last few numbers one feels that the record of the years gone before has been equalled by that of last year. A large number of papers have been read, and one sees that not only the immediate activities of the Rand itself have received the attention of members, but papers have also been read describing metallurgical processes and mining operations in other parts of the world, showing that members take an interest also in what is happening elsewhere, and are ready to apply the knowledge and experience gained there in local practice. In addition, I am glad to see that the question of health and safety has also received attention; for example, in a paper read by one of your distinguished Past Presidents, Mr. W. Cullen. Another sphere of the activity of your Society is that devoted to excursions, which I think very laudable indeed. It strikes one in going about the mines here that one often sees some person carrying out some process which, if he took the trouble of going to the neighbouring mine, he would see being done very much better and at less cost. If he would take the trouble to walk half a mile he would be able to benefit by the experience. Excursions such as you make do a great deal to open the eyes of others to the merit of visits to other mines. In the past, the general activities of the Society have been confined to the gold mining carried on on the Rand. This, of course, is really due to the fact that the gold mining industry on the Rand comprises 99% of all mining and metallurgy in the Transvaal; though it seems as if that is going to be changed in the future. If we look at the records of this country we see that base metals are gradually beginning to play an important part. Next to the gold we have tin and copper coming in. As far as tin is concerned, it is progressing very rapidly, but copper more slowly, as there is only, as yet, one company with any appreciable output, though with greater facilities for transport I hope that this may be improved upon. Copper is one of the most interesting metals metallurgically, and if the copper industry expands in the future many of the members of this Society will be able to show what they can do in this line. Then there is another metal to which I am sure members will pay increasing attention in the future, and that is the king of metals, iron. You are probably aware that the iron resources of this country are enormous, though they are lying fallow and virgin. You have, no doubt, seen or heard of the statement made by the Prime Minister of this Colony a few weeks ago, in which he stated that the Government intended to start in conjunction with the C.S.A.R. a melting furnace for working up scrap iron. There seems to be a great deal of misapprehension about this matter. It is not that the Government and the C.S.A.R. intend to be iron masters. No, the facts are these. There is a large amount of scrap being exported from this country. We are sending it away to Europe at prices which hardly pay expenses. It is treated there, and probably sent back to us at a very much enhanced price. Gentlemen, why cannot we do that ourselves? We have had proposals in the past, and several companies have even made a start, but what has become of their efforts? Nothing. They have liquidated and disappeared. In order to remedy that state of affairs and act as pioneers, the Government and the C.S.A.R. have determined to put up a plant. Whether this will be in the form of an electrical furnace or of another kind has not yet been decided. It may be said that this is a very paltry way of attempting an iron industry, but working up scrap is only the beginning. The iron industry is a much larger question, so much so that it is not one that the Government can lightly undertake. I think the Government will be sympathetic with any efforts made to start blast furnaces in this country, but it has to proceed very carefully. There is also a question which I would like to bring to your notice which concerns the mines of the Rand more immediately. It is that of ventilation. I am glad to see Mr. Albu nodding his head at me. He is one of the pioneers as far as ventilation is concerned. When you have encouraged it and put it into your mines I am sure you will never regret it. It has to come within a very few years. Your deep level mines will never be worked satisfactorily without the best ventilation. We have been talking about efficiency the last year or two, and its relation to £ s. d. When you have ventilation which contributes to the efficiency of your workers, you will get back your £ s. d. over and over again. In addition to the £ s. d. you will also have the health of your workers improved. Gentlemen, I have said enough about the past activities of this Society, and have shown also some of the things which may be tackled in the future and, therefore, I think I have given you reason enough to drink the toast I am going to propose. I give you the toast of "The Chemical, Metallurgical and Mining Society of South Africa."

The President, in responding, said: Mr. Kotzé, Mr. Mayor and gentlemen, it is one of the proudest moments of my life when I stand before you as the President of the Chemical, Metallurgical and Mining Society, to return thanks for the toast which has been so ably proposed by Mr. Kotzé, and by the way in which it has been responded to, I can see it has been an acceptable one to you. Gentlemen, why do we meet on these occasions? We meet not for the purposes of science but from a social point of view in order that we may all come together round the festive board and meet one another and learn to know one another better, and so promote the feeling of friendship and cameraderie, which, as members of our Society, we must feel for one another. The only chemical part I can see about the meeting is that we keep going the chemical process of life by a due assimilation of food and drink (strong or otherwise as may be the taste of the individual concerned). By this means we are carrying out a chemical work. By having these annual festivals men who are separated from one another in the course of their daily avocations, it may be by 50 miles, come together and get to know one another. Further we have the opportunity of meeting and becoming acquainted with those gentlemen who favour us with their presence as our guests. I am sure there is a great deal of good done both to the Society and ourselves by these meetings. The Society had its birth practically at one of these festivals, and I well remember that evening. I daresay there are a good many amongst you who also remember it, and I think it is quite fitting that we should keep up the memory of the birth of our Society by an annual festival, and so perpetuate the friendship which existed amongst the few members of which it was originally composed and cause that feeling of friendship to extend. I am very pleased to see that so many members have turned up to-night to aid in the dissemination of that friendship we all desire. With the increase of the mining industry our Society has also increased, and we have now a total membership of about 1,080—rather over I think. I think that is very creditable, and think we may consider ourselves the strongest Society in South Africa. The great difficulty is to find anyone connected with the mining industry who is not a member of our Society. Our members are spread all over the world. We have members in Central America, then across the world to the eastern archipelago. Lately we have elected a member in Japan. There is not a very wide stretch of country anywhere which has not a member of this Society in it. In connection with the progress of the mining industry I should like to refer to the report for the month of March this year for the Transvaal. It is compiled by the Government Mining Engineer's Department and shows that fine gold to the value of 2½ millions sterling was produced for the month. The number of mines actually crushing was 111, and the number of stamps dropping was 9,705, and also 128 tube mills. The development of secondary crushing by tube mills, in spite of the objections raised when they were first introduced only a few years ago, is really wonderful. Tube mills seem to have come not only to stay, but to multiply. The report goes on to show that the total white and coloured men employed amounted to over 200,000, and taking the amount of fine gold produced, and the number of individuals employed, it will be found that for every unit employed there is a monthly production of £12½ sterling of gold, and I do not think there is any country in the world which can show a return like that, considering the number employed. But although the progress has been great, finality has not yet been reached, I am sure there are many hundreds of claims still to be opened up as mining propositions which will give further employment for present and future members of the Chemical, Metallurgical and Mining Society. Although the values of these claims may not be so high as the more favoured portion of the reef, yet with lower working costs and better extraction, and by the exercise of our opportunities, I have no doubt whatever they will give a satisfactory return to those who have the courage to find the capital for their exploitation. Leaving the subject of the noble metal gold for the moment, as Mr. Kotzé has stated, I think no room whatever exists for doubt that in the future we shall have a large base metal industry in the country, and that the development of these base metals will become a great and important factor in the development of the country. Base metals have been found and continue to be found in various parts of the country, not only common base metals but some of the rarer metals. The possibilities of the country are unknown as yet, and I think it will take a long time before we do know them. With regard to the iron industry, it is cheering to hear that some steps are being taken with regard to this, although, of course, we can scarcely look upon the re-melting of scrap as being an iron industry. Enormous deposits of iron ore exist in the country, and these deposits must be used. We cannot go on as a country continually importing from oversea, and as we require iron and steel in the industry, something must be done to provide it on the spot. Also with regard to other industries. It would appear from the papers that there is a possibility also of a mineral oil industry in the country, and going on to another subject, when the Union, which I think all are so earnestly looking forward to, has been carried out, and when we can get the co-operation of the several states in destroying the various pests which beset agriculture, and also assist in eliminating as far as possible the sicknesses which affect horses, cattle, and sheep in this country, in which work, I hope, the chemists connected with our Society will take a part. When we have found out the nature of the various soils in the various parts of the country, which differ greatly, and what are the right and proper materials to use as fertilisers, it appears that then we shall have an enormous agricultural industry also. It means that the country will arive at that desideratum of every country that it will be "self-supporting." While touching upon agriculture, I feel that the greatest tribute must be paid to the arduous work which has been performed by those gentlemen who have been connected with the Department of Agriculture. As our Society embraces chemistry in all its branches, I am sure that we, as a Society, should welcome amongst our members any of those gentlemen connected with the Department of Agriculture who would like to join us. A certain amount of blame has in some quarters been placed upon the Society for adding the term "Mining" to its title, but I am sure that anyone who reads our *Journal* and has read the interesting and attractive mining papers that have been placed before us and which have led on to other subjects such as explosives, and the resultant gases formed, which all tend to the study and betterment of the conditions of the worker underground, will admit that the addition of "Mining" to our Society has been amply justified. I should like also to mention that at a recent meeting of the Society an alteration of one of its rules was made, by means of which it is now possible to admit as student members those young men—lads practically—who have been recently taken on in reduction works, and not only those in the reduction works, but those who are working underground, with a view to the practical study of mining. We felt that we should try and do something for these lads. Our rule heretofore has been confined to certificated students of some college or mining school. Well, we thought it was better to broaden that and make it possible to admit all those young fellows who are working in the reduction works and underground so that they may have the benefit of receiving our *Journal*, attending our meetings, mixing with the older members of the Society, and gaining information in every way. Gentlemen, I do not think I have any thing more to say. I will again thank my friend Mr. Kotzé for the kind and able way in which he has proposed the toast of the Society, and you Gentlemen, for the cordial way in which you have received that toast.

Mr. W. H. Dawe (Vice-President, Transvaal Chamber of Mines), in proposing the toast of South Africa, said: You will understand that I undertake the duty of submitting this important toast with a feeling of considerable responsibility. The honour has devolved upon me in the absence of the President of the Chamber of Mines, Mr. Langermann, who, being a South African by birth, would no doubt have revelled in the opportunity which this occasion would have offered him. I have used the expression "South African by birth" because whilst admitting the claim of birth right, I wish to press another claim—one which I possess, and one which in itself would give me also the right, had I the necessary eloquence, to propose this toast. It is the fact that South Africa is my home. I have lived here for many years, the best working years of my life so far; my chief interests are centred in this country, and last, but far from least, the majority of my best and most valued friends are here. Many of you are similarly placed, and would not hesitate to acknowledge your love for South Africa, and that you feel the "grip" with which it holds its people. Our loyalty is not of less account because we were born in other parts of the Empire: it is rather of greater value, seeing that it is the outcome of experience. The love and loyalty which its children bear to South Africa is as natural as the air they breathe, and if we, its adopted children, offer like love and loyalty, I claim that it is a tribute which our country can proudly accept. For that reason I am glad for the moment that I, one of its adopted children, am called upon to sing its praises. And what is the reason of this affection; the meaning of this hold which South Africa undoubtedly has upon its people? It is that in addition to all its natural advantages, it is a country in the making, and as such, exercises an irresistible fascination over all who wish to live in the fullest sense of the term. It is this which appeals to all men of action; to those who wish to take their part in the making of a nation. Let us endeavour to see what claims it has on others—to the historian, the statesman, the lover of nature, the sportsman, men of business, and men of science. The student of history follows its story with enthusiasm, for as the pages unfold he reads of progress made against insuperable difficulties. It is a tale of continuous struggle—now against natural difficulties, now against a savage foe; sometimes of efforts crowned with success; sometimes of disaster, but never of defeat. And right through the story we have a spirit of heroism which kindles the blood and makes us proud of those of our fellow-countrymen who bore the heat and burden of the day. And what of the lover of nature? Can he name a spot more beautiful, or more generously endowed by nature than the Cape Peninsula? He will tell you of its manifold and varied beauties and its never ending variety, crowned by the mountain which invests the scene with majesty. And this beauty is only at the gate of a country which can show many other scenes of grandeur. If you have spent a day in the veld you will have gained an experience not easily forgotten. You have realised a feeling of solitude which at first almost filled you with dismay. The silence seemed unbearable, but gradually you realised that the world around you was instinct with life, the loneliness was forgotten, and, bathed in the glorious sunshine, you felt the joy of living. These are but two of the many scenes that could be easily recalled, and I ask you whether the most ardent lover of nature would not be satisfied? And if you have not had these experiences, you can understand my meaning by reading two books which will give you perfect word-pictures of such scenes—"The Story of an African Farm" and "Jock of the Bushveld." And now, gentlemen, to turn to the more practical features. Can the man of action ask for greater opportunities than those offered him in South Africa? There are openings on every hand, and if each effort be not crowned with success, is it not probable that often the reason of failure lies with the man himself? There are great difficulties to be overcome, and no industry has more of these to contend with than agriculture. Yet I venture to say that few countries offer greater opportunities to the agriculturist. If he be properly equipped with technical knowledge, capital and experience, he will make light of the obstacles to be overcome, and meet with a surprising measure of success. We have recently had an invaluable object lesson in regard to the progress which is being made in this great industry, and we appreciate to the fullest extent the assistance which the Governments of South Africa are rendering towards its development and progress. And I think perhaps you will not object to my mentioning the name of one, whose individual efforts have been of such incalculable value, and that is the name of the President of the Witwatersrand Agricultural Society—Mr. Lionel Phillips, who has done so much for its advancement. And what opportunities has this country given to men of science? It is a question which the majority of those here this evening can very readily answer; in fact, the answer is provided by the fact of the existence of your Society. You will agree with me that in assisting to build up and perfect this great gold mining industry, the scope provided for your ability is practically unlimited. As the representative of the Chamber of Mines, I may be allowed to express the appreciation which is felt in regard to the invaluable work the members of your Society have already accomplished, and as gratitude is an appreciation of favours to come, you will conclude that we expect great things from you in the future. I am glad to be able to refer to the great change which has taken place in the prospects of the industry during the past year. The feeling of depression has given way to confidence in the future, and we see expansion on every side. But, gentlemen, it is well to remember the causes of this change. In the first place, we had to put our house in order. For over three years this work of re-organisation has been strenuously carried on, and no doubt we can claim with the most beneficial results. It would be merely wasting time for me to tell you of the possibilities resulting from the reduction in working costs; you are as well informed on that point as I am. But I question if these possibilities would have been generally recognised had it not been for two other important features—the wonderfully good strikes made at different parts of the Rand, and the inauguration of a policy of amalgamation of companies. The first dissipated all doubt regarding the value of the reef in depth; the second gives stability, by extending the lives of properties and making it possible to effect greater economy in working. So you see the improved position of the industry is the result of organisation and co-operation, and I would ask you whether that which is applicable to an industry cannot be applied with equal success to the needs of the country. We are all anxious that South Africa should take its place in the Empire, and there is only one way to attain that end. Is the dream of Union to become a reality, or will an effort be made to perpetuate the present state of affairs? In the past it has been the misfortune of South Africa to suffer from distrust; distrust between the two virile races who live here side by side; and often distrust of the Government caused by acts which greater knowledge would have prevented. Surely, the time for that has passed, and we are ready to recognise that, if we are to build up a nation worthy of the name, it cannot be accomplished if preference be given to any section of the community. Union of the country is worthless without unity of aim and purpose. The Transvaal and its Government has been in the van of the movement in favour of Union; it has advanced no more favourable claim, and it has been ready to sink all considerations as to whether the Transvaal would give more than it received. If any were inclined to count the cost they were few in number, and they were unheeded. We kept in view the wider policy and recognised that the whole of South Africa was to be considered. The benefits of a united country far outweighed mere local considerations, and as far as the Mining Industry is concerned, we are willing that our interests should be transferred from the keeping of a Government which understands our requirements to a Government, the majority of whose members will in all probability be unacquainted with them. So much have we been willing to give, and I venture to think that this has lead to the supposition that we are willing to put all consideration aside to secure Union. But if I understand the feeling of the people of the Transvaal, this supposition is entirely fallacious. We have made a condition, and that is equal rights, and we have no intention of yielding on that point. That Union should not be accomplished would be disastrous, but that it should be accomplished by the preferential treatment of any section of the community would be ten times more so; in fact, it would be impossible. We have every confidence in our delegates, and we know that our interests are safe in their hands, but more is expected from us, and every opportunity should be taken, during the short time which must elapse before the Convention meets again, to express in unmistakable terms, that in their insistence of equal rights, our delegates have the undivided support of the people of this Colony. We most sincerely hope that the wiser counsels will prevail and that Union of our country will be accomplished on an equitable basis, and then, and then only shall we realise our strength and the glory of our heritage, and take the place in the Empire which is rightly ours.

Sir Percy FitzPatrick, M.L.A., replying to to the toast, said: Progress, of course, is intimately connected with the toast to which I have to respond, and which Mr. Dawe has proposed in a particularly eloquent and striking speech. It is a good many years since I had the privilege of speaking at one of your Society's dinners, and I do not know really at how many I have been present, but they have all been very good dinners and excellent gatherings. It has always been a pleasure to meet you upon these occasions. I think it has often been said, and no doubt it will be said with increasing frequency and force, that this Society in some way typifies the condition of South Africa to-day. You can look back upon the history of South Africa. You know the period in which everything was left to Nature and the few willing but greatly handicapped, earnest, uninstructed people who tried to make headway against immense obstacles. It is easy for you to look back and see when the great change began in this country. That is the time from which you can date the influence of science upon South Africa. That is where you ought to date your starting point, because South Africa can be taken in two chapters—either in periods or in two chapters, or in parallel developments; two different kinds of development and progress. What has made South Africa to-day is not the old order or the natural conditions, but the newer development and the influence of science that has been brought to bear upon it. We have had two developments just as we have had two races. Personally, I do not see why they should not work together in perfect harmony—not in rivalry, but as necessary complements of each other. I am as sure as I stand here to-night that we are going to have that. Of course, we have obstacles in front of us, but I think we are accustomed to obstacles. I think South Africa would lose half or three-quarters of its charm if we had no obstacles. I am sure, too, that in the pursuit of your professions you would find it a very tame thing indeed if you had no difficulties to overcome, and, what is much more serious, you would find that nine-tenths of you would be without profitable occupation. I remember a few years back talking to a gentlemen who is prominent and successful in the growing of fruit trees. I said to him, "what a pity it is that we have such a number of diseases in this country." He said, "that is why I came here." I said "why?" He said, "by the Lord I should have no chance in a country where every fool could grow fruit." And so I suppose you take it that way yourselves. If every fool could develop the mining industry in all its branches with the same ease as clever men, why should you be here? You would be having 5s. a day. That is the way it goes throughout South Africa. We are finding and we are going to find that the able men, the earnest men, the hard-workers are going to reap the reward which they richly deserve, and, not only that, but that the country is going to reap the reward.

The Obstacles to Union.—It would be very difficult for me in responding to the toast of South Africa to avoid the subject that has been occupying your minds to some extent, and my time entirely, for the last six months, that is the work of the National Convention. And I should think probably that you would think there was not much use in getting me upon my legs if I were not going to talk about a subject I know something about, and in which you are interested. I am in a position of greater freedom and less responsibility than some of the gentlemen whom you would like to hear upon the subject. We have our obstacles there. You have only to read the daily papers, which sometimes record and sometimes create those obstacles, to find that we have still some serious difficulties before us. I do not know what your politics are, and, to tell you the honest truth, I do not much care—at the present time—especially as I am not a candidate for election. You possibly know my politics better than I know yours. On the assumption that you do, I suppose that you would agree with me that, being in a position of no responsibility and great freedom, my duty would be to slam-bang the Government of the day as hard as I could. Still I am not going to do it; not only because we have declared a kind of truce—I hope it will last long enough to satisfy everybody—but also because we have a very serious object in common, and I really do not see how one can mix up two things which are conflicting, one of which is very important—the union of South Africa—and the other relatively unimportant—our party politics. They can wait, and when the time comes I suppose we will all take the matter up just as keenly, if necessary, as we did before. In the meantime we have a big national object before us, and that is to achieve the union of South Africa. With that apology I would like to say a word upon a subject on which I am not called upon to speak at all, which discretion prompts me to leave alone, and that is the Mozambique Treaty. The responsibility of that rests primarily, of course, upon the Transvaal Government; next and almost equally upon the Governments of all the self-governing Colonies of South Africa, because they were amply consulted, and they were parties to it before it was arranged. Subsidiarily and thirdly, a certain responsibility rests upon every member of the National Convention, and in that subsidiary position of responsibility, of course, I stand. But I have heard so much of this plea, alleged or true, "it was not me, Sir, it was the other boy," that I am rather tired of it, and I am prepared to accept my share of responsibility and something more, because I think it essential that we should face this thing, and have a clear understanding, as it stands in the way of a satisfactory Union of South Africa. To that extent, and for that reason, I think we might all take our share of responsibility. I would like without going into details, for which of course, none but the Governments or the Government is responsible, to put before you one or two considerations. They are serious considerations for this country. The first consideration is this: That the Delagoa Bay railway route is 100 miles shorter and infinitely better in the way of gradients—being 1 in 60 against 1 in 35 of the Natal route—than any other. That is a crucial point to be borne in mind, and you need not pay very much attention to trial trains or reports so long as you have that fact well in mind.

Our Trade Alone.—The second point is this, that the trade in dispute and being divided is our trade. It is *our* trade. It is nobody else's trade. Nobody in the world has the slightest right to divide that trade but ourselves. When this question was raised, we knew that this particular trade of this particular area of South Africa, was selected out of all the rest of South Africa to be divided up. No other trade is interferred with, only this. We accepted that position for the good of South Africa. We were then faced, or the Governments were faced, with two alternatives—either to hand over the whole position with the *Modus Vivendi* to the Union Government, or to make a fresh treaty, because both the Natal Government and the Cape Colony Government in the last Convention, when they met, had as the first item upon their programme the denunciation of the *Modus Vivendi*. We knew we had those difficulties to face. Look at it impartially and as people who really believe in union, and who are prepared to trust their partners in South Africa. I am in that position. I am prepared to trust my fellow-colonists, in Natal, in the Cape Colony, the Free State and the Transvaal, absolutely. As long as we have equal rights I am prepared to trust them absolutely. If you take up that position there was only one logical course, and that was to hand over the whole position to the Union Government. Why did we not adopt that course? Well, I tell you that the Cape Colony and Natal had upon their agenda before union was discussed, the denunciation of the *Modus Vivendi*, and it has been in the papers for years. Their first plank was the denunciation of the *Modus Vivendi*. I don't blame our fellow-colonists, because they had strong convictions, and they knew they were losing the trade. Putting myself in their place, I say they have done well for their Colonies, and the censure which has been heaped upon them is undeserved. But what puzzles me in looking at this is, to find from any given critic an answer to this question: "Whom are you defending?" I can't make it out. When you hear them on one point—the defence of the Transvaal—they will take up the case of Natal. When you have beaten them on that they will take up the case of the Cape Colony. If you have to run the entire gamut of South Africa, defending each one, there must be something to be said on the ground of fairness of compromise for this Treaty, if they are all dissatisfied. They are dissatisfied. Even Portugal is dissatisfied.

The Alternative. — The alternative to this Treaty would be to hand over the whole position with the *Modus Vivendi* to the Union. The Transvaal was willing in the last resort to do so, but the suggestion was not accepted, and I will tell you why. Natal—and I do not blame them—felt they could not take this risk, knowing that they were only a small minority in the Union, and they would have no security to offer to their people, as to the trade they would have. That was No. 1. Then you had the Transvaal position. What was to happen to the whole of the Eastern Transvaal, to the farmers who supplied the mines on that side, the coal mines, the tin mines, the properties which Mr. Kotzé referred to, and the possibility of development on that side—what was to happen to that if the Union took the line of shutting up Delagoa Bay and developing through Union ports, which is a legitimate national line to take up? Could you have carried that in the Transvaal? Impossible, absolutely impossible to carry it. The Transvaal people were going into union, making great sacrifices, as you know, but large sections of them—you cannot blame them—were not prepared to gamble their whole future, and to be completely ruined by the Union. They could make a reasonable reservation in defence of their very existence. The farmers of the Transvaal would not have agreed to it. The people who are developing the mining industry, in coal, tin, and other metals, in outlying districts would not have agreed to it. The merchants of this country would not have agreed to it, and the people helping in this mining industry, whose labour supply might have been cut off, would not have agreed to it. By one sweep, by one act, you would have united every section in the country against union. Now, I ask you, do you blame the Transvaal Government for having tried to arrange for a period of ten years only, something which will enable them to carry union? You cannot fairly blame them. I am not a member of the Transvaal Government, I am not a supporter of the Transvaal Government, but I tell you honestly you cannot stand up and blame these men for taking that line. It has been said that better terms ought to have been obtained from Portugal. Now, I ask you again to consider this position rationally. To every attempt made for a reduction of the terms, the Portuguese had this answer: "You have your rights under your treaty;

exercise them." To exercise them meant to continue the *Modus Vivendi*, to denounce it as soon as union comes into force, lose your labour supply at once, shut off your trade, put a pressure upon them in a year's time, suffer all you can suffer for a year before you can bring any pressure upon them; and in the meantime not merely the Transvaal suffering through its industries, but the Cape Colony losing all its trade, Natal coming down from 30%, as it was a year ago, to 22%, as it is to-day, and to 10%, as Col. Greene the other day said it would be, before Union could deal with the position. What prospect was there to carry that? It came to this, that in order to satisfy our neighbours and to give fractional protection to our own people, which would enable us to carry union here, something in the nature of a new treaty had to be established. That was necessary to safeguard conditions which the Transvaal and its neighbours alike agreed were essential to the carrying of union. But here the Government of the Transvaal had to face another difficulty or complication, and you must do it the justice to recognise it. The new treaty was a condition precedent to union, yet union itself was no certainty, as the position to-day amply testifies. Thus the Transvaal had to negotiate a treaty whose terms would in the one case be satisfactory to the Union if union were accomplished; and, in the other case, be workable for the Transvaal if through the rejection of the draft Constitution by any of the other Colonies, the Transvaal should be driven to either complete isolation or partial union. Failure to consider and provide for this last grave alternative, which the position to-day shows is no imaginary one, would have left the Transvaal powerless to defend its rights and legitimate interests. We should have been either compelled on the one hand to accede to any fresh demand from any other Colony or, on the other hand, reduced to relying on the goodwill of the Portuguese. Would the Parliament and people of the Transvaal have accepted that? There is no doubt about the answer. When you come to criticise the details of that treaty, you will find that there is ample power for the future Union Government to defend itself, and I think you will find at every turn the answer to us, when we wanted better terms, was "exercise your present rights." An impasse! Now I hope I have not said so much as will embarrass anybody else, or put myself in the position of saying that I firmly believe in every detail of that treaty. I have told you what my responsibility was. I have put myself up for criticism and I shall get it, I know. But I have done it with the object of making an appeal to reason, and justice, and commonsense. It is a relatively unimportant thing. This is a compromise which was insisted upon by our neighbours equally with ourselves. This is an arrangement which does not reflect discredit upon Natal, or upon the Cape Colony delegates, or the Transvaal. We have given a good deal. Natal has secured a great deal for her people, the Cape Colony has secured for East London and Port Elizabeth probably double what they have to-day. But we do not grudge it to them. Let those who want to find fault with this recognise that it is as much a compromise as anything else, and that the Union Government will be all-supreme and all-powerful and able to deal with this from the Union point of view in the Union interest in ten years' time. We have, it is true, departed from the principle laid down, a most important principle, that the carriers of this country—the railways—should be conducted practically at cost. That is of supreme importance to the development of the interior. We have not abandoned the principle, but we have in a measure suspended the operation of it for ten years, in exactly the same way as we have suspended the operation of equal rights for ten years, by giving to Natal and to the O.R.C. more than they are entitled to in the Assembly, by giving them equal representation in the Senate. We have, it is true, by this treaty, restricted the power of the Supreme Parliament to deal with Union affairs, but so we did in deference to the Cape Colony's difficulties in the matter of the native vote. But in all these things please remember that where we have had to depart from the great principles which we have endeavoured to establish, and which we have established in the Constitution, we have only done so for a fixed period. The great principles are there, and after ten years, which is not a long period in the life of a country, all these handicaps or lieus upon the power of the Government, or the principles which we value most and which are most essential, will disappear, and then you will have this Union founded upon equal rights, and with a Supreme Parliament able to take care of Union interests. Perhaps I have made too much of this difficulty, the treaty, but it was with the object of showing you and appealing to you to believe that it is a relatively unimportant one. Get it out of the way and see to the essential.

Bedrock Principles.—The essential thing is the Union, and to its accomplishment the all-essential thing is the principle of equal rights. Gentlemen, whether you are South African born, as some of us are here, or, as Mr. Dawe said, whether you belong to the other class who by choice have made South Africa their home, and whom we value as much as we value ourselves, you all know that the trouble in South Africa has been due to this—that there has been rivalry between sections and races as to which or who should have the power. When Mr. Dawe was talking about the mining industry and the cause of the present revival, I was thinking that the real reason why you are having better times to-day, and why you have more confidence in the future, is that you know the industry is upon a sounder basis. It has come to a sounder basis, because those who had control of the industry—those who had the driving force, or who were inspirers of movement and enterprise—went right down to the root of things. You cannot alter the size of your reef or the gold it contains, but you can alter the working costs. They went right down to bedrock to see what was wrong and what was assailable ; and so we have done in South Africa, and in the search for this we have found that one cause of the unstable position was the lack of principle in the distribution of power and the everlasting struggle as to who should be top dog. Until that was remedied, there could be no peace—there could be no co-operation. Energies were spent and wasted on rivalry, instead of being utilised in joint action. It is for that reason that I value equal rights. I have said it before, and I say it again, that until we have it firmly fixed all else is trifling. Every sacrifice is worth while if you get peace between the races, based upon equal rights. The struggle to-day—the struggle which we see before us—is a very natural one. It is a struggle which you have seen many times in history. Those who had the power have always been unwilling to part with it. You or I would do the same if we were in the same position. It is no use judging harshly. Look at it calmly. You will find it is human nature. At the present time, fortunately for us. we have men of both parties—men of both races—who realise that that must not be perpetuated, that therein is suicide and ruin for South Africa, that that must be changed.

Look for Results.—Credit neither to one race nor to another, neither to one party nor to another. Do not give credit. Look for the result. There are men who are not in the same position to speak as I am to-night, working as earnestly as I am and as my colleagues are to put this matter right. Help them. Help them by standing firm to your principles. Help them by maintaining this—that you must have the principle of equality in order to establish peace. Gentlemen, if you were to offer me this Union with the least little bit of advantage given to one race or one class, I would say, " take it away." There is the seed of discord—the beginnings of a festering sore that will corrupt the whole body. Get it out. I do not care myself, whether an impartial tribunal shall give to a country district, more than to a town district. It does not trouble me. The impartial tribunal, I have confidence, will do justice according to the circumstances of the day in each case. But I will not see in our Constitution the brand of inferiority put upon one section or one race. May I tell you how important this is? I will give you the sequence in this matter at the Convention, and I hope that that is not a breach of confidence. I do not see how it can be. At any rate, you have not got from me what I have done on different questions—you are not going to. I am not going to tell you what anybody else did. But I think that, in defence of the principle of equal rights, you are entitled to know the way we settled matters. First came the language question. We settled it upon its merits. I refuse to make it a bargaining counter. We stand by it, and I stand by it. Whatever is done on any other question, I stand by it. We then settled the question of equal rights. Now, mark what happened afterwards. We then settled the powers of Provincial Councils. Do you suppose that if we had not got equal rights we would have consented to hand over the whole of the Transvaal affairs to the Central Parliament in which we did not have equal rights? We would not have done it. Then we settled the division of trade. Why should we give away our trade? Do you think we should have done that had we not secured that in any settlement of this question in the future we should have equal rights, man for man—would anybody else? It was little enough to ask. We accepted it, because we had equal rights. Then we had the matters of finance. You know that the profits tax brings in nearly one million pounds a year. You know that shares in the diamond mines make it considerably more—say, another half-million. You know the bewaarplaatsen, and the mining rights not disposed of, are increasing every day and will total up to over two millions a year, and perhaps a good deal more, in the course of a few years. I ask you, do you suppose we should have handed over those resources if we did not believe we were going to be fairly treated and have equal rights? Never! Then we come to the question of the Senate, in which existing Parliaments can, if they choose, exercise the rights of existing majorities and put their own complexion upon the Senate for ten years. Do you suppose that we would have surrended that right to another party had we not got equal rights ourselves to come into force in full ten years? Never! Then there was the question of the division of the capital. All these questions were settled after the question of equal rights had been settled.

The Foundation Stone.—I believe, gentlemen, that those who are trying to alter what we consider to be a fundamental condition, an absolute essential of union, do not realise that this is one of the first stones laid, that it was the foundation upon which the other settlements were built, and that they cannot draw that out from the foundation without bringing down the whole structure. I read a letter the other day from an anonymous writer—I wish for his own sake the writer had put his name to it—dealing with this question. He took a rather gloomy view, a much gloomier view than I take, and his concluding works were: "In Bloemfontein there yawns the open grave of equal rights." When I read that, I thought of something I had read before: "I have digged a pit for my friend, and have myself fallen therein." There is a grave dug and opened in Bloemfontein, but it is not the grave of equal rights. That has yet to be dug. You will never live to see the tombstone that commemorates the burial of equal rights in this country. That grave is going to contain the mouldering bones of that accursed thing, racialism. That is what we are going to bury at Bloemfontein. Do not be downhearted. It is the principle we have stood for, and nothing will shake us upon it. Mr. Dawe said that no union is worth while without equal rights. Your material interests are at stake. But I brush these considerations aside. It is because equal rights is the basis of peace and the future of South Africa, and because the harmony of the two races is going to be built upon it, that I value it above all things. You are all deeply interested and concerned in this, and you feel it deeply. I know there has been some anxiety. I want to tell you, so far as my judgment goes, so far as my determination goes—and I am certain I am speaking with equal certainty on behalf of my colleagues—there will be no wavering upon this point. I say this in no spirit of hostility, and with no desire to stir feeling against our fellow-Colonists and our fellow-South Africans in different parts of South Africa. We know that they have a lot to learn. We have a lot to learn about them. They have a lot to learn about us. They have mistaken ideas as to what the development of the Transvaal is. They have not had the same opportunity as we have had of mixing race with race and knowing the leaders on both sides, the intentions on both sides, and the policies on both sides. They have not rubbed the angles off that we have. Be a little patient with them. I believe that most of them are absolutely sincere, but they do not realise how little they have to fear, and how much they have to hope, from the principle of equal rights, which will give them peace between the races and the prosperity of our own beloved country, South Africa.

Dr. Jas. Moir, in proposing the toast of "Our Guests" and "Kindred Societies," said:—Mr. President, Mr. Mayor and Gentlemen,—It is my pleasant duty to-night to propose to you the hospitable toast of the evening, that of the "Guests" and "Kindred Societies." At this time of night, however, I feel myself like a certain wedding guest of old, who was of opinion that any sort of rubbish will do all right at the end of a celebration of this sort, and consequently I regret that, owing to my disabilities, you will have to do without eloquence. Now, Gentlemen, there is nothing new to be said about guests in general, and I am not going to talk commonplaces, and as for our guests in particular, the main thing is that they are good fellows, and, besides, most are really not guests at all, having the good sense, except in a few instances, to be members of our Society. The chief appeal in this toast is that it is a personal one, and not an abstraction like its predecessors. As for the Kindred Societies, the only thing to be said is: "More power to their elbow." Our critics have said that there is no room, with our small population, for so many technical societies as we have; that their work overlaps, and so on. This, I take it, must have been in the nature of a "brain-wave," arising out of a very bad attack of "co-operatie." And this leads me to what I specially want to say to-night, which is that what Johannesburg seems to need at present is not fortitude or rectitude—with all due respect to the Municipal motto—but thoroughness and education—I am almost tempted to say primary education. The bald fact is that, so far from this being a specially brainy place, in reality the most of the brainy men of Johannesburg, I believe, could be squeezed into this dining hall without serious inconvenience; and, as a matter of fact, our Society always flatters itself that this annual gathering does really attract most of the talent of the place. I am not saying that there are not plenty of people in the Transvaal with aspirations after education and culture—the attendances at Prof. Raleigh's and Prof. Fisher's lectures show this—but I do really doubt whether culture has made much actual progress so far. Take the Press of this town, and we know the high mission of the Press nowadays; how it has come, through the spread of knowledge, to occupy in all civilised countries the position once held by the pulpit—well, really, you would think from our Press that this isn't a civilised town. Will it be believed that one of our dailies, in trying to quote Latin, actually spelt Agamemnon with a small *a*; that the same paper last week said that Marion Crawford before his death was reading Pl*u*to's dialogues (shades of Plato and Socrates!), that others spell "vaccillation" with two *c*'s, as if it had something to do with smallpox, and speak of a station*e*ry tramcar?—all mistakes which no schoolboy should make. Take our

Municipality next—and I hope the Mayor will pardon my mentioning some of the delinquencies of his predecessors—look at its tramcar signs: "Cars stop here by request." I suppose any ordinary un-municipal person would have thought that cars stop by switching off the current and by application of the brake instead of "by request," and could have told them that it should be "on request." Then we have the amusing advertisement, which used to emanate from the Municipal Pound: they used to excite us by announcing in the papers, "the following is confined in the pound: a bay gelding." I have already mentioned the town's 'swagger' motto, "Fortiter and recte," and I will now only say by way of a joke that it is so modest and unassuming that I am only surprised that while they were at it, the authorities did not go the whole hog and claim, say, piety and good looks as well. Judging from the date of the motto, I suspect that *fortitude* is a covert reference to the Chinese dispensation; and as for rectitude it smacks rather too much of a death-bed repentance after spending five millions. However, I see from the menu before me that in talking like this, I am guilty of pot calling kettle black. Our Society also has a swagger Latin motto, "Capaci occasio." Now, Gentlemen, this motto does *not* mean, as you might suppose at a festive moment like this, "a chance for the capacious"; it merely means "an opportunity for the competent"—so you see that this Society has symptoms of 'swelled head' like other people. I'm afraid it's a very catching complaint in this climate. Still, I remember once Mr. John R. Williams saying that our aim should be to try and equal our possibilities, and if we combine this with our present motto, we get at the root of the matter, something really worth aspiring to, namely, thoroughness in everything, so far as we possibly can. Well, Gentlemen, I have blethered long enough, and you will say that I have treated our friends who are outside the scientific circle — for, of course, we ourselves are the salt of the earth, the elect—in a very unfriendly and cynical way, something in the style of Mark Twain's Essay on Friendship: "The holy passion Friendship is of so sweet and steady and loyal and enduring a nature, that it will last throughout a whole lifetime, if not asked to lend money"; therefore I want to say that naught has been set down in malice, and that I am sure that there is quite a lot of potential culture here which only wants to be organised and made accurate and thorough. Gentlemen, I give you the good health of our Guests, coupled with the name of Sir Thomas Price, and prosperity to our Kindred Societies, coupled with the name of Dr. Temple Mursell.

Sir Thomas Price, in responding, said: Mr. President, Mr. Mayor, Dr. Moir and Gentlemen. It was my misfortune not to catch the whole of the eloquent speech in which Dr. Moir submitted to you the toast of the Visitors and Kindred Societies, but from the applause with which the observations he made were greeted, it was quite clear as to the consideration he was extending to the visitors and kindred societies. I heard just two words and they impressed me. One was "good looks," and the other was "fortitude." I shall leave both of these to my friend, Dr. Mursell, to respond to, as he is much more qualified to deal with the question of good looks than the present speaker. May I say to you, Mr. President, that you have given us a good dinner, you have treated us as comrades and brothers, you have given us a most enjoyable evening, you have given us an opportunity of listening to eloquent and important speeches, and as a spokesman of the visitors, they would not wish, and I certainly shall not be so ungrateful as to inflict a speech upon you for the purpose of removing some of the impressions which have been created by the addresses they have heard. But I do desire to express to you on behalf of the visitors our admiration for the work you have done, and of the extraordinary extent to which the people of South Africa are indebted to the members of this Society for the progress and the unique position in the world which this country occupies, and particularly with regard to the industry with which you are so much associated. May I say that instead of your talking about what you will do, instead of shouting about what you have done, you have occupied your time in making the industry, in making the work with which you are associated take absolutely and quite clearly the first rank in the whole world. You have succeeded in accomplishing things which were considered even a few years ago impossible of accomplishment. You have done things which all over the world elsewhere, they have failed to accomplish, and we in South Africa owe you a deep debt of gratitude for what you have done. One thing I would desire to draw your attention to is this. I think you will expect me to refer to the railways. With regard to the railways, you may take it that petrol traction is going to take a not unimportant share in rendering certain service on the railway and I appeal to you as the Chemical and Metallurgical Society to see to it that you will provide us with something that can be produced in South Africa to work these motors, without our having to go elsewhere for petrol. You have done so much in other directions that it seems to me to be a little thing for me to ask you to do, namely, to find the necessary material to supply us with petrol for working that particular service required on the railway. The other thing I ask you to do is this. You have heard about the position which base metals are going to occupy in the welfare of this country. I entirely agree with all that has been said, and I do feel that I have opportunities of knowing a little of what the prospects are in that direction. My appeal to you is this, that you will see to it that instead of our having to send base metal ores in a concentrated form out of the country, you will provide the means necessary to see that these base metals shall go out of the country in a merchantable form, so that you can use the base metals in your own country to begin with, and send your surplus oversea after treatment in South Africa. We will make a bargain. You have got very low rates for concentrates to-day, produce these metals in merchantable form and the speaker will be one of the first to help you along with regard to low rates, to enable the merchantable article to be exported, at a profit to this country. One gentleman says "Union will settle that. Well, if I know anything about union and the position the Transvaal will occupy in the union, and, may I say, if I know anything of the people of South Africa outside the Transvaal; if you produce the article in merchantable form, I do not think the people of this country will stand in your way so far as railway rates are concerned, in getting these, base metals in the form of ingots exported. Let the public come along and see the quantities of scrap the railways and mines are producing. It will, however, be one of the most valuable articles you have, to combine with some of the ores you have got in this country, and which my friend, Mr. Kotzé has so much turned his attention to. We will then, when it can be so used, be quite ready to increase the rates on scrap metal. Gentlemen, I am sorry I am intruding upon you unnecessary railway questions. I am here replying on behalf of the visitors, but the temptation was too great to answer the questions put to me as a railway man.

Dr. H. Temple Mursell (Vice-President, Transvaal Medical Society), who also responded, said: "I feel some diffidence in getting up to respond to this toast. In the first place I have to endeavour to the best of my ability to represent the President of the Society, which I am more particularly associated with—Dr. Porter—who, I sincerely regret, is not at the present moment in South Africa, or he would undoubtedly have made a point of being here; and I regret his absence extremely on my own behalf because I think he is the wittiest and most eloquent speaker that my profession manages to produce. Another reason why I have a more than usual diffidence is because there is an ancient and absolute rule in my Society which forbids the presence, as guests, of any other than members of my own profession at our annual dinner. Consequently I feel that I have, to a certain extent, obtained a most excellent dinner under false pretences. But I have reason to believe that largely through Dr. Porter's own request that rule will be abrogated, and I trust that at our next dinner we may have the pleasure of seeing your representative there as our guest. Now I have always been led to believe that your Society is the premier scientific society in this country. I see that remark has met with your full approval. I do not wish for a moment to cast any doubt upon it, for you have this unique advantage amongst scientific societies, you can appeal to the man in the street. There is an idea at the back of his mind, rightly or wrongly, that however abstruse, however scientific your debates or papers may be, that you have one ultimate aim in view, namely, the reduction of working costs. Now that, of course, appeals to him at once. So far as I am personally concerned when, from time to time, I have endeavoured to read some of your transactions, I have been more or less mystified; especially when I read such abstruse things as Dr. Moir's effort to produce something new about the atomic theory. It is many years ago that I had the misfortune to study something about the atomic theory and I have completely forgotten all about it. As for the Society which I immediately represent, I am afraid it does not appeal in the smallest degree to the general public. It is regarded with a considerable amount of suspicion and distrust, the prevailing impression being that we are met to talk at regular periods and confess to each other the mistakes we made since we met last. I can assure you that this is absolutely wrong. We never reveal any mistakes at all. There is one idea firmly fixed in the mind of the man in the street, and that is we make no endeavour to reduce working costs. I freely admit there is not the smallest necessity, because so far as my own small experience goes, the general public does that most efficiently for itself. If they get into the slightest difficulty there are no working costs at all. I am supposed to return thanks to the toast of kindred societies. As I have only the faintest idea of the number of societies to which you claim kindred, I cannot mention them individually, but I wish to associate them with my sincere expression of thanks for the splendid hospitality you have shown us to-night, and to thank you from the bottom of my heart for the privilege of being allowed to be present here."

The Mayor (Mr. C. Chudleigh): Gentlemen, I dare say that you have all looked at the toast list and felt that the items on that toast list have been completed. Yet I am bound to say, there is one very important one which does not appear on the list, and that is the toast of your worthy President. I should just like to say a word or two in reference to your Society, and particularly to your President. I am told that he was one of the few who some fifteen years ago assisted at the inauguration or creation of your Society. He has been a member all that time and, as he said—and I am sure you will agree with him—it was one of the proudest moments of his life when he stood up to reply to the toast of your Association. I dare say you all feel proud of your Society and of your President. I feel, as a commercial man, somewhat out of place when I see so many scientific men and, as far as I can see, all connected with the mining industry. I am a commercial man, and I am proud to be one. Still I am going to lay claim to belong to the mining industry. When Mr. Dawe was proposing the toast of South Africa, he referred to the veld. I may say I once spent nearly two years on the veld, and I travelled once in a bullock wagon more than 69 days, and I can assure you, Gentlemen, the charm of the veld has never left me, and when I say that in 1887 I was digging for gold, I think you ought to accept me as belonging to the mining industry. Therefore, I feel that in proposing the toast of your President I am proud to do so as a man connected with the mining industry, and I also feel proud to be present here to-night as representing the town. On behalf of the town I desire you to accept my heartiest thanks.

The President, in response, said:—Gentlemen, it only remains now for me to thank our worthy Mayor, Mr. Chudleigh, for having so kindly proposed my health. As I told you before, it was an exceedingly proud moment when I stood up to respond as your President to the toast of the Society and the kindly way in which the toast was proposed and received would, if I were not such a modest man, tend to produce that terrible disease, which I believe our medical friends would designate, "oedema elephas capititis," as there must be an "itis" in these days. However, Gentlemen, I do not propose to suffer from "swollen head." I thank you, Mr. Mayor, very heartily for the kind words you have spoken and you, Gentlemen, for the kind way in which you have received the toast.

The singing of the National Anthem brought the gathering to a close.

Obituary.

The following deaths are recorded with much regret:—

Mr. ARTHUR WARREN KENDALL PEIRCE, M.Am I.E E., M.Am.Soc.Mech.E., Station Superintendent of the Victoria Falls Power Co, Ltd. (Germiston Branch), died on the 13th April from ptomaine poisoning and malarial fever, contracted during a recent visit to Zululand. Mr. Peirce, who joined the Society as an Associate in July, 1902, was formerly Consulting Electrical Engineer to the Consolidated Gold Fields of S.A., Ltd, and General Manager of the General Electric Power Co., and the news of his sudden death came as a great shock to his many friends.

Mr. GEORGE PERCY WOODVILLE ROFE, Assoc. S.A.S.M., Metallurgist, Riverina Gold Mine, Western Australia, the death of whom was notified recently, was admitted an Associate of the Society in July, 1905.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

RUSTING OF IRON.—"The rôle played by CO_2 furnishes the foundation for a series of experiments the results of which are summated as follows: 1. Iron will rust under conditions in which CO_2 is absolutely absent. 2. Air containing 15% CO_2 under the same conditions is only twice as active as air absolutely free from CO_2. It is not likely or possible that the small amount present in the atmosphere exerts any action on the process of rusting. 3. The degree of attack is dependent on the manner of introducing oxygen (bubbling of air, exposure to diffusing air, etc.). The introduction of oxygen by bubbling air through the solution is about twice as active in promoting rust as the simple contact of the surface of the liquid with the air. 4. The rusting is dependent on the partial pressure of the oxygen in the atmosphere above the liquid immersing the sample. In an atmosphere of pure oxygen the attack is three times as strong as in ordinary air. Pure CO_2 causes no peculiar rusting. It acts as any acid by dissolving with the evolution of hydrogen. True rust only appears on the introduction of oxygen Free oxygen is necessary for rusting, the presence of the oxygen combining with hydrogen as water playing no part in the action. Iron is an extraordinary sensitive reagent for the determination of oxygen dissolved in water. It is not, however, available for quantitative determination because the loss of weight of the iron is partly due to oxide in each of the two states, the proportion of ferrous and ferric not being constant. Rusting may be prevented by removal of oxygen from water. Evacuation and boiling are not technically possible in most cases. Even when removed, a coating of oil will not prevent reabsorption of air. With this knowledge of corrosion it may be said that the velocity of rusting is proportional to the distance of the metal from the surface of the liquid in contact with the atmosphere; to the area of this surface of contact; to the rate at which oxygen is bubbled through the solution: to the partial pressure of the oxygen in the atmosphere above the liquid; to the rate at which the fluid in which the iron is placed is renewed; to the proportion between the surface of the iron and the volume of the liquid it is in; a mathematical discussion is given of these ratios. The action of hydrogen peroxide in various concentrations allowed of no particular conclusions being drawn.

In general, it was proven that iron rusts less, and other metals more, when placed in contact with a metal lower in the e.m.f. series than the iron, in a solution which ordinarily rusts iron. The opposite is true when in contact with a metal above itself in the series. A sample of iron in contact with copper, rusted 25% more in the tap water of Charlottenburg, and 47% more in artificial sea water, than when placed in these two solutions alone. Contact with nickel increased the attack on the iron 14 to 19%. Cast iron in contact with wrought iron protects the wrought iron at its own expense. The wrought iron showed 28% less corrosion in a solution through which air was bubbled, and 50% less in a solution in contact with quiet air. In general, it was found that iron in contact with identical metal mechanically treated in various ways, showed a difference in potential between the various samples. When overheated and non-overheated metal are placed in contact in water, the non-overheated metal is more strongly rusted than when placed therein alone."—HEYN AND BAUER.—*Electrochemical and Metallurgical Industry,* Jan., 1909, p. 33. (A. McA. J.)

COLORIMETRIC METHODS.—In one modification of this method a solution of sodium plumbite, made by dissolving 60 gm. of lead acetate and 90 gm. of caustic soda in 1 litre of water, is added to the sulphide solution to be tested and the colour matched by adding a standard solution of lead nitrate to a solution containing an excess of sulphuretted hydrogen or of sodium sulphide, the two solutions being placed in cylinders of the same diameter filled to the same depth in the usual way. The assumption made is that when the depth of colour of the two solutions is the same, they will contain the same quantity of lead sulphide. This is by no means the case. For example, solutions containing sodium sulphide equivalent to 0·0006 gm. of potassium sulphide were treated with 2, 5, 10, and 20 c.c. of the alkaline lead solution and the colour matched by adding a solution of lead nitrate (0.15 gm. per litre) to a solution containing 0·015 gm. of sodium sulphide (total volume in each case = 60 c.c.). The quantities of the lead nitrate solution required were 15·1, 11·8, 12·4, and 10·4 c.c. respectively. Again, solutions containing 10 gm. of potassium cyanide and 0·0015 gm. of potassium sulphide were treated with 50 c.c. and 5 c.c. of the alkaline lead solution, and matched against a solution containing sulphuretted hydrogen equivalent to 0·425 gm. of potassium sulphide. The quantities of lead nitrate solution required were 15·1 c.c. in the one case, and 21·0 c.c. in the other. The quantity of sulphuretted hydrogen or of sodium sulphide used in the comparison solution also affects the result, for example, solutions containing 0·0006 gm. of potassium sulphide and 1 c.c. of the alkaline lead solution in 60 c.c. were matched against solutions containing 0·015, 0·03, and 0·075 gm. of sodium sulphide; 14·8, 12·4, and 10·9 c.c. of the lead nitrate solution were required in the three cases. The difference is not so large when different quantities of sulphuretted hydrogen are used, but it is in the same direction. Finally, the depth of colour is not the same in solutions of sodium sulphide and of sulphuretted hydrogen. two solutions containing sulphide equivalent to 0·033 gm. of potassium sulphide were prepared: the one contained sulphuretted hydrogen, the other sodium sulphide; 10 c.c. of the lead nitrate solution added to the first gave the same depth of colour as 13 c.c. added to the second.

In all these experiments the solutions were made up just before use, with water which had been boiled and cooled out of contact with air. It follows that the depth of colour of the lead sulphide is affected by the nature and quantity of the other constituents of the solution. Comparisons of colour are only possible so long as the lead sulphide remains colloidal. Linder and Picton's work on colloidal arsenic sulphide (*Journal of the Chemical Society*, 1895, 63, and 1905, 1914) suggests that the colloidal lead sulphide consists of complex molecular aggregates $n\,(m\mathrm{PbS.H_2S})$, in which the hydrogen is replaceable by metals. The colour would then depend on the values of n, m, and on the nature of the sulphide combined with the lead sulphide, which would, in their turn, depend on the nature of the substances present in the solution. Hence a reliable colorimetric estimation can only be made when the two solutions have the same composition. This condition is fulfilled in the ingenious method described by G. W. Williams (*Journal of the Chemical, Metallurgical and Mining Society of South Africa*, 1905, *6*, 170; this *Journal*, 1906, 137). Two equal quantities of the cyanide are dissolved in boiled water, the sulphide is removed from one of them by shaking with lead carbonate and filtering, the same quantity of alkaline lead solution is then added to each and the colour matched by adding a standard solution of sodium sulphide. Another advantage of the method is that atmospheric oxygen acts approximately equally on the two solutions, and therefore the error due to this source is eliminated to a great extent.

A sample of potassium cyanide tested by this method gave 0·0309 and 0·0296% of potassium sulphide, whereas the direct titration method described above gave 0·0246 and 0·0240% of potassium sulphide, or, after correction by multiplying by 1·25, 0·0037 and 0 0300% of potassium sulphide.

A few experiments were made with the method described by W. Feld (this *Journal*, 1903, 1068), in which the sulphide solution is distilled with excess of magnesium chloride in a current of carbon dioxide, the sulphuretted hydrogen evolved being absorbed in a known quantity of iodine solution acidified with hydrochloric acid. The apparatus used was that previously described (this *Journal*, 1904, 244) with the addition of a set of potash bulbs containing iodine solution and a second U-tube containing a little $N/10$ sodium thiosulphate. In order to see whether any oxidation of sulphide takes place in the distillation, a solution of sodium sulphide was (I.) precipitated with pure zinc chloride and the sulphur in the filtrate estimated, after oxidation, as barium sulphate. II. Another portion of the solution was run into a boiling solution of magnesium chloride and boiled until free from sulphuretted hydrogen, the residual sulphur being estimated as barium sulphate. III. A third portion was treated in the same way after adding a solution of 10 gm. of pure crystallised NaCN.$2H_2O$ in boiled water to it. The results were :--

Sodium sulphide used.	Residual sulphur calculated as sodium sulphide.		
	I.	II.	III.
gm.	gm.	gm.	gm.
0·162	0·0013	0·0024	0·0037
0·013	0 0005	0 0002	0·0001

It appears, therefore, that the sulphuretted hydrogen can be expelled quantitatively, and provided that a sufficiently long train of absorbing vessels is used, it is easy to absorb it in the iodine solution. For example, in one test 0·0736 gm. of sodium sulphide was distilled and 0·740 gm. found from the iodine used.

Summary.—Of the three methods discussed the direct titration of sulphide by means of lead nitrate is the most rapid and simple ; suitably corrected it is sufficiently accurate (for quantities of potassium sulphide under 0·1%). G. W. Williams' modification of the colorimetric method is more accurate, but it is slower, owing to the necessity of using boiled water and checking the strength of the standard sodium sulphide solution at frequent intervals. (A solution containing 2 gm. of potassium sulphide per litre, for example, lost 28% in 18 hours.) W. Feld's method is the most accurate, but it is far too slow for works' purposes, and it can only be used by a skilled operator."—THOMAS EWAN.—*Journal of the Society of Chemical Industry*, Jan., 1909, p. 12. (A. McA. J.)

ORIGIN OF THE DIAMOND.—"In an article by Mr. S. Meunier, published last year in *Science au XX Siècle*, the origin of the diamond is discussed, especially from the point of view of meteorites, which have played such an important rôle in the study of this problem, leading to the well-known experiments of Moissan. After examining meteoritic iron from the Canon Diablo, Moissan concluded that enormous pressures had intervened in the formation of this meteorite, like all others, and that this was the key to the question. Consequently he concluded that spontaneous production of heavy pressures will produce artificial diamonds. His idea was correct, observes Mr. Meunier, although the method invented by Moissan did not give, perhaps, the great degree of pressure he imagined. However, there is something much more important omitted by Moissan, because he had not sufficiently studied meteoritic iron, which is the fact that everything in their structure and composition tends to prove that meteorites are not produced by fusion. Several proofs can be given. In the first place, fusion absolutely disorganises meteoritic iron, and the slowest cooling is incapable of restoring its initial structure, and analysis demonstrates that there are very unequally fusible substances in contact in these meteorites. For example, iron and nickel alloys, fusible at over 1,200° C., are very delicately arranged around a core of iron sulphide, which could not resist 400° C. From these analyses and observations, it results that meteoritic iron is produced by concretions and not solidification of fused matter. This completely alters the aspect of the mode of formation of the diamond and the studies undertaken to solve this problem industrially. In fact, experiments have demonstrated that the elements of meteoritic iron can be reconstituted and assembled by the action of properly selected fumes at red heat. Thus, according to Mr. Meunier, theory points to the possibility of obtaining diamonds from the reaction of hydrogen gas under pressure on carbon chloride fumes. This question of pressure is very difficult. At the ordinary pressure amorphous carbon, or at most graphite, is produced, and the plant to compress gases during their reaction, which must be of long duration, has yet to be invented."—*The Mining Journal* (London), Feb. 13, 1909, p. 217. (A. R.)

LIQUID AMMONIA.—"A review was given of the devices now in use in this country and abroad, and it was pointed out that a source of error inherent in some of them was the moisture contained in the atmosphere, which could not be excluded from the sample. The new apparatus excludes this error entirely and its accuracy was demonstrated by repeating the experiments of Lange and Heffter (*Chemische Industrie*, 1898, p. 2), with liquefied ammonia gas of the greatest purity, and also with mixtures of this ammonia with known quantities of benzole, pyridine, alcohol and water.

The liquefied ammonia gas used in these tests had been made from sulphate of ammonium previously purified from all volatile carbon compounds. For this reason it was certain that no carbon compounds like benzole, pyridine, alcohol, etc., could be present, and the limit of water was ascertained (by treatment with metallic sodium and measuring the hydrogen evolved) to be less than 0·002%.

The average of five analyses made in the new apparatus of this ammonia showed it upon evaporation to leave a residue of not more than 0·00175%, consisting principally of water, iron oxide and lubricating oil, while Lange and Heffter, using Urban's method, have worked with liquefied ammonia gas leaving upon evaporation 0·2% of a non-volatile residue.

Evaporation test of pure liquefied ammonia gas containing known quantities of benzole, pyridine, ethyl alcohol or water :—

Substances present.	Liquid NH_3. gm.	Addition. gm.	Addition evaporated. % by Urban.	% by Frerichs.
Benzole and pyridine	99·0	1·0	—	15
Benzole	98·5	1·5	41	59
Pyridine	99·0	1·0	33	24
Ethyl alcohol ...	99·0	1·0	20	17
Water	98·9	1·1	17	11
			Used ammonia leaving upon evaporation.	
			0·2%	None.

The limit of accuracy of testing with the new apparatus has been ascertained as follows :—

For water to	0·002%
For alcohol to	0·006%
For pyridine to...	0·009%
For benzole to	0·110%

The new process of Dr. Frerichs (United States patent 905,415, Dec. 1, 1908) for making pure liquid ammonia is about to be tried on a large scale. Its essential features are as follows : Commercial sulphate of ammonium is heated in a still to about 200° C.—a temperature sufficiently high to volatilise the carbon compounds in the salt and to convert the nitrogenous-organic compounds into sulphate of ammonium by reacting with the sulphuric acid of the ammonium salt. No considerable quantities of ammonia gas are thereby liberated from the salt. To facilitate the process of purification, an air blast may be introduced in the still. After this purification process is over, the temperature is raised to 350°, to get the reaction

$$(NH_4)_2SO_4 = NH_4HSO_4 + NH_3.$$

If the temperature is further raised to 400° C., the reaction is quickened. No ammonia is destroyed, but the acid sulphate of ammonia is decomposed into pyrosulphate of ammonium and water:—

$$2NH_4HSO_4 = (NH_4)_2S_2O_2 + H_2O.$$

The resulting pure ammonia gas is then worked into the desired products, while the acid sulphate of ammonium and pyrosulphate of ammonium resulting from the operation are dissolved in water, and the resulting solution of acid sulphate of ammonium is utilised for absorbing volatile ammonium compounds and impure ammonia gas coming from crude ammoniacal liquors during the initial distillation thereof or otherwise obtained. This completes the cycle of operation, the sulphate of ammonia thus obtained being treated as before."—F. W. Frerichs. —*Electrochemical and Metallurgical Industry*, Feb., 1909, p. 63. (A. McA. J.)

METALLURGY.

The Leading Nation.—"The Germans have made the greatest advance in economic metallurgy of any nation in this world during the last five years. They have utilised their by-products to a greater extent than any other manufacturing nation. Not only that, but they have developed their mechanical appliances with reference to manufacturing to an extent that no other nation has, and they have developed their quality to a greater extent than any other nation during these past five years. In other words, manufacturing in Germany five years ago seems to have had a complete renaissance, and they have advanced very much more rapidly than any other nation, for the reasons I have given. 'For two reasons: the first, the very excellent technical education of their metallurgical engineers in Germany: the second, the most important, the labour conditions in Germany as compared with the conditions in England. I think the labour conditions in England are the worst of any of the great manufacturing countries of the world. . . . Germany is rapidly adopting the same methods and blast-furnace practices that we are, with reference to economies in handling the material, but they have gone much further than we have in the introduction of the use of their waste gases. Germany was the first country to use waste gases for the development of power, which makes a very large saving in the cost of pig iron. We are rapidly adopting it, but, to tell the truth, we are following Germany in that respect."—Charles M. Schawb, Committee of Ways and Means, Washington.—*Engineering and Mining Journal*, Dec. 26, 1908, p. 1,267. (A. R.)

Lead Acetate in Cyanidation.—Mr. C. M. Eye writes to the *Mining and Scientific Press* to call the attention of other cyanide workers, especially of those using lead acetate in their work, to a case where, in a mill in charge of the writer several years ago, litharge was substituted for that salt, to good advantage. The material being treated was a concentrated tailing re-ground to pass 50 mesh, the greater part of the slime being removed before treatment by agitation in vats 20 by 20 ft. Thus the material agitated was largely fine sand, containing much pyrrhotite and a little arsenopyrite and chalcopyrite. The stirring action was very strong.

Soluble sulphides formed during treatment required the addition of at least 2½ lb. lead acetate per ton of solids in the charge. As this formed a heavy item of expense, William Magenau, the chemist of the mill, experimented with other lead salts and found that litharge, added to the charge in the proportion of 1½ lb. per ton, was equally effective. As litharge cost only half as much as acetate, it effected a reduction in cost for this item to 30% of the former figure. Besides, it was easier to use, simply requiring to be weighed and sifted into the agitating charge.

This substitution may not be generally applicable, nor in some cases even possible, but it has occurred to me that it might be used to advantage in tube-mill work, adding the litharge with the feed. This would be the logical place to add the litharge when grinding in tube-mills with cyanide solution, which, in my opinion, is likely to be the practice of the near future."—C. M. Eye.—*Mining and Scientific Press*, Jan. 9, 1909, p. 82. (K. L. G.)

Milling Costs.—"If a close margin of profit is conducive to reduced costs and increased efficiency, it can be introduced into any mill by assuming that the ore is bought from the mine on an arbitrary margin, the basis of purchase established being maintained throughout, whether it show loss or gain. For instance, in a certain mill the average experience for the preceding year shows a net profit of 40% of the average gross value of the feed. Then establish as a basis of purchase from the mine, say 38% of the assay value of the feed on date of delivery to the mill. This will leave a 2% margin of profit under conditions existing for the previous year, and will establish a standard for comparison. But fluctuating conditions will make it interesting to keep the balance on the profit side of the ledger, and the figuring can be made as closely as if the ore were actually being purchased in this way. For example, a lead concentrator treating 1,000 tons per day and working under the above assumption with lead at 5 cents per lb. and silver at 55 cents per oz., would make a profit of about $2,000 per month. With the prices reduced to 4 cents per lb. and 50 cents per oz., the extraction and operating costs being the same, the result would be a deficit of about $500 per month. These figures leave a vastly different impression from the customary ones involving gross values of hundreds of thousands of dollars.

With the above assumption for a starter, the following suggestions might be applied for arriving at milling costs and efficiency. They apply, in this case, to a lead-silver concentrator.

1. Get the dry weight of ore milled each day.
2. Arrange the feed-sampling apparatus so that the assay of the feed represents the actual contents of it, rather than an abstract figure on an assay report.
3. Have the spouts carrying the concentrate discharge into a trunk launder for each class of product, and let these discharge into their respective bins in such a way as to admit of easy time-sampling.
4. Have time-samples taken hourly of the different classes of concentrate from the above launders and the total sample for 24 hours of each weighed and sampled for moisture and assay in such a way that the least time will elapse between the end of the sampling period and the assay results.

5. Calculate the gross value of the marketable mineral content of the feed and of the product each day at the market rate for that day. The ratio gives the extraction accomplished.

6. Assume a constant as cost for the ore at the mill, say, a certain percentage of the assay value, as a standard for comparison.

7. Figure the actual profit from shipping the different grades of product in No. 4.

8. Find the net daily profit by subtracting No. 6 from No. 7 and the daily cost of operating from the result.

9. Have each machine in the mill arranged for sampling its feed and products separately by time-sample, if possible.

10. Number each machine and determine the cost of operating it for 24 hours.

With the above data in shape you can, with practically the same amount of labour as is now devoted to sampling and assaying grab samples of tailing and products from which no tangible results can be obtained, get actual results in dollars and cents each day, showing very closely the profit from operations and the efficiency of the plant.

The test-sheets enable you to have a number of tests made each day on individual machines, depending as to number upon the capacity of the assay office and the sampling force. These tests should be made systematically, but not in regular rotation, so that the operatives, knowing their machines to be next in order, could have them in apple-pie order for the occasion. If these tests were made impartially, regardless of the conditions under which the machine is running, some interesting data could be gathered as to the losses from machines working badly temporarily. These data, I believe, cannot be gathered from 'experience.' The above system would undoubtedly tend to increase the efficiency of millmen."—R. S. HANDY.—*Mining and Scientific Press*, Jan. 23, 1909, p. 156. (A. R.)

MINING.

TAPERED TIMBER PROPS.—"When props were set tight and the roof began to weight and move, this movement tended to tilt the prop slightly, with the result that some portion of the base of the prop was not resting on anything. The foot was really broken in sections, and when the last yielded, the prop kinked uphill. Indications of the inequality of this force might sometimes be found in the lid, one portion being compressed or crushed more than another. His belief was that there was too much timber at the base of the prop to allow it to 'rock' with, or lend itself to, the roof movement.

It was, however, quite a different matter with either the tapered or the sharpened prop, and was probably one reason why the latter lasted much longer in inclined measures. The question of tapering bars was particularly important, as that district was specially suited for its exploiting. It was questionable whether any district in England was put to so much expense for road repairs. It was not unusual to find that the majority of bars broken or breaking on some of the levels were broken through side weight, and when ripping and re-timbering was being done the amount of dirt drawn off was nothing like proportionate to the damage. Some ten or twelve years ago, the writer heard of a system in vogue at a local colliery where men were employed by the butties at the week-end, loosening the ends of bars by cutting the sides free. In a working stall in the Barnsley bed, or Nottingham Top Hard coal, which was 5 ft. thick, the roof was barred, the bars being ordinary 5 ft. props split in two and spaced from 4 ft. to 4 ft. 6 in. apart. Packs were kept built up to the timber and the holding was usually 5 ft. deep, kept if possible close up to the tub. By this method the weight was transferred along by the tapered props on to the face and proved of material assistance in breaking down the coal. The floor was hard and the roof heavy and tender, and the pack side prop was usually the old face prop of the last web or buttock.

The objection to the tapered prop had been raised that it was a question of economy rather than safety, but he contended that the tapered prop was more economical and efficient, a point on which both masters and men agreed. The less need there was for timber to be changed, the better for everybody, for a whole prop was a good prop, and by adding to its period of use, they added to their own safety. Every stallman he questioned was greatly in favour of tapering, giving as reasons that there was less re-setting of timber, much less trouble with the roof owing to falls in the stalls, and that the weight operated more on the coals. Another objection raised was that packing was the best method of supporting the roof and carrying the weight on to the face. One surely agreed with this, and in no district in England were the rules regarding packing more strictly enforced than in the Barnsley bed area. There, one of the principal differences between the ordinary and tapered prop was apparent. The one being a rigid body resisted the natural subsidence of the roof, and the other assisted in the movement, more weight being thrown on the face in consequence, without the roof being broken. Another objection was that the roof weight was intermittent and not regular. One could not altogether agree to this, as to the best of one's belief there were two weights. One the working weight, which was regular and consistent, the other the breaking-down weight in which case the roof broke down in the wastes. This was the intermittent one, known as the weight coming on. If the working weight was not regular, one felt inclined to ask, 'What force is it that tightens props, sometimes very rapidly?' In Sawyer's 'Accidents in Mines,' they found him saying: 'There is at times a tendency to set props slack, relying on the tightening action of the roof and floor to fasten them, as they are then in some cases not so soon broken.' He was not advancing this method of timber setting, but was directing attention to it for two reasons. First, what was the tightening action of the roof and floor? He thought they might conclude that this was due to the working weight previously mentioned. Secondly, how did they account for props which were not set tight not being so soon broken? Apparently the only explanation was that some little play having been allowed for roof subsidence, the life of the prop was consequently prolonged.

In considering the application of tapering to some of the longwall workings in North Staffordshire, the conditions tending towards its successful use in the Barnsley bed might be mentioned. There was a hard floor, the measures were flat, and the previously mentioned system of barring the roof in the stalls was important, as the props, with the bar as a connection, exerted a steadying influence on each other and carried the roof weight more uniformly. Taking the most important item, that the measures were flat, one was aware that the timber was not mauled to anything like the same extent as in inclined measures, but with a little extra care much timber might be saved, firstly, by being set on the proper dip, and, secondly, by being set straight. There was, he believed, scope for its successful application in some of the many seams worked on the longwall system in that district.

A few advantages of tapering were that it was more economical, more efficient, and safer. It assisted in breaking down the coal, and a better roof was kept. The props were frequently re-used, whilst in some collieries they were taken out of the pit and re-tapered for another and thinner seam than the one where they were first used. There was less liability to stalls being off when the drawing shift commenced, and the timber was more easily withdrawn. Lastly, the wood was not spoiled to the same extent as in the ordinary prop, and could be cut up for sleepers, lids, etc."—P. Horan, *North Staffordshire Inst. of M. & M. E.—Iron and Coal Trades Review*, Feb. 5, 1909, p. 193. (A. R.)

MISCELLANEOUS.

Practice and Science.—"Amongst our members we number at least three distinct grades, the manufacturer, the scientist, the engineer. In some cases the three grades may be enrolled in one individual. Such cases are rare, perhaps, but in many at least two of the above named attributes are the qualifications of the same associate.

The daily occupation of the manufacturer is to handle various metals, and convert them from their raw material condition to some form of manufactured article. In the process of so doing he employs that knowledge which has been handed down to him by his forbears, or which he may himself have gained by a lifetime of study and experience of the peculiarities of such metals and alloys as his work may bring him in contact with. Each day he accumulates fresh practical experience and knowledge, and yet from time to time he meets with some new phase which puzzles him; or happens upon some discovery of value; the latter often some very small matter which would at first sight have been passed over as of no importance, but which by accident has attracted attention. As often as not the cause and effect are noticed and taken advantage of in practice, but the reason why is never discovered, and so it happens that the road thus shown is not explored.

It is here that the scientist pure and simple comes in, or should come in; given results, cause and effect, the scientist, if he persistently devotes himself to the task which is offered him, will probably eventually arrive at the why and the wherefore of the matter. He has the necessary time; his vocation, as a rule, is, if I may so describe it, embowered in that cloistered seclusion which allows of consecutive thought and reasoning out of obscure and difficult subjects; whilst to any man immersed in the stress and toil of the twentieth century commerce, such things are almost impossible.

Again we have the engineer. In the pursuit of his professional work, the engineer has to select such materials as are best calculated to meet his requirements.

In so doing he is largely guided by the experience of those who have gone before him; or, if he be a pioneer in any special direction, by such data as are available of the nature and properties of various materials which may or may not be suited to meet his needs. Having selected his materials, he proceeds to construct his machine or whatever he desires to produce. If he has chosen well and wisely, he may be satisfied, but this is by no means always the case; he may find that his choice has not resulted as he expected, troubles arise and failures occur which were not anticipated. He is most likely at a loss to account for these, and after investigating as far as the information at his disposal will allow him, he most likely arrives at the conclusion that there is something wrong with the manufacture of his materials, quarrels with the maker and adopts someone else's manufacture, possibly experiencing a like result, and eventually abandoning that class of material altogether, and using something quite different—to everybody's detriment.

Most likely, if that engineer and manufacturer had placed a little confidence in each other, and carefully discussed the question before embarking upon the use of the material, it would never have been employed, and much trouble and expense would have been saved. Or if, after the trouble occurred, they had called into consultation a competent and qualified scientist, he would most likely have been able to locate the cause of trouble.

Hitherto the engineer has made unto himself what may fairly be described as his own law, and has made arbitrary specifications of what tests the material he wishes to employ shall stand, and then having obtained these desiderata, proceeded to use the material; if subsequently failure occurred, he has at once placed the blame on the maker, who most likely has in his turn attributed the failure to improper usage, or the selection of a material not properly applicable to the requirements of the situation. Neither side being prepared to acknowledge the possibility of error on their own part, matters arrive at a deadlock, and the matter is never satisfactorily cleared up.

If an umpire in the guise of an independent scientist, fully qualified to undertake an investigation, has been called in, and the question thoroughly sifted to the bottom with the aid of knowledge and information in the possession of all parties, probably valuable light would have been thrown on the subject; and information gained not only of use in this specific instance, but also to engineers and manufacturers in general and the world at large.

Laboratory work is of immense value, and without it nothing can be done, but it is not sufficient for the scientist to demonstrate in the laboratory and lecture room. Science must be reduced to practical form for everyday use, before it can be made serviceable in manufacture. Many large establishments have their own laboratories and scientific staffs, and in such cases it is a comparatively simple matter to develop a practical application from information gained by laboratory experiment. In many instances, however, this is not the case, and the practical worker has to depend on the outside scientist, and needs guidance in not too elaborate a form.

It would be practically impossible to run a mill or foundry with heating and cooling curves, microphotography, slip of crystal, etc., as the only guide. These are all necessary points of knowledge upon which to base a practical works practice, but to get full value from these it is necessary to bring science into touch with practice, and for the man from the laboratory to assimilate a personal practical knowledge of the processes of manufacture; then he can teach. One cannot make the average workman into a scientific expert, but one can instruct him that he must do certain things to obtain certain results; and one can see, more or less, that, having been told, he does them. Knowledge is power, but power often runs to waste, and the knowledge of non-ferrous metals has hitherto run much to waste; chiefly through jealous fear of teaching a trade opponent something, whereas he has probably long since discovered it for himself. Each individual has some cherished bit of knowledge, some trade secret which he hoards carefully. Perhaps by sharing it with others, he might impart useful information; but by an open discussion and interchange he would, almost for certain, learn a dozen things in exchange for the one given away. General increase of knowledge would give general improved practice, most likely a larger use of the materials in which a manufacturer is interested. Each would give a little, and receive a hundredfold in exchange. No man knows everything: even the cleverest has lots to learn yet; and after all, with equal opportunities of knowledge, it will still be the best man that will best apply that knowledge; if members have courage and confidence in themselves, they need not hesitate to exchange their knowledge with that of their colleagues or even their adversaries."—SIR GERARD A. MUNTZ.—*Institute of Metals.—Page's Weekly*, Jan. 22, 1909, p. 160. (A. R.)

Reviews and New Books.

THE WORLD'S GOLD.—By L. DE LAUNAY, Professor at the Ecole Supérieur des Mines. Translated by Orlando Cyprian Williams, with a Special Introduction by Charles A. Conant. 6s. (London: William Heinemann.)

"He examines the geological occurrence of gold within the limits of industrial conquest, and estimates the possible extent of the total supply which the world contains. He then proceeds to consider the countries where gold has been mined, and how much of the original store may yet remain, with a view to ascertain what countries will furnish the supply of the future, and how long it is likely to be maintained at its present high figure. This brings into consideration also the improvements in mining and metallurgy, which, as the Rand has rapidly taught us, have a powerful effect on tha available supply. Having thus established his basis, Mr. De Launay proceeds to examine the position of gold; its uses in the past and at the present day, the possible supply within a limited time, and the demands which that supply may have to meet, and thus arrives at the final inquiry as to the effect on prices and on our financial system of the movement under consideration. The conclusion at which he arrives is that the rich deposits are necessarily highly local, and are so easily exhausted that it is only by pushing forward the boundaries of civilisation into the unexplored parts of the world that fresh discoveries of 'easy' gold can be hoped for, and that the future production depends upon the poorer ores. With regard to these the supply is increased as the expense of production is lowered, or as the value of the metal increases. In fact, so far as man is concerned, the supply of gold in the earth may be regarded as practically unlimited, and society will always have as much gold as it is prepared to pay for. Mr. De Launay is able to satisfy himself that there was in fact in the nineteenth century, up to 1873, a reduction in the purchasing power of gold, but the correlation of this with the variation of supply is not clear. A rise in the price of commodities does appear to have set in about 1895, and after the interruption of the Transvaal war was continued up to 1906. The fall in price after the American panic seems to have confirmed Mr. De Launay in his general theory, and, indeed, to have led him to the conclusion which brilliantly winds up the book—that there has been no excessive production of gold, and that in spite of the increase of the last twenty years the world is now suffering from scarcity of the precious metal. But could not the movement equally well be used to support the theorem that an apparent scarcity of gold follows as a consequence on an excessive production? The result of the inquiry to which the author has addressed himself as to the effect on prices and on our social economy of the increased supply of gold, is that the problem is insoluble. There are, no doubt, people, and those not mere circle squarers, who will not be ready to accept this answer, but no one will grudge the time spent in following Mr. De Launay through this brilliant and fascinating lecture. One of its most interesting features is the ingenuity of the method by which a basis of calculation is arrived at in cases where, as with regard to the total supply of gold in the earth, statistics are impossible, or, as in the case of the amount of gold reduced into the possession of man, they are demonstrably untruthful. Put forward with the extraordinary lucidity which is characteristic of French scientific work, this is a vivifying study in economic speculation, and even those who are dissatisfied with the lecturer's conclusions will be well pleased with their entertainment. It remains to say that the translator has done his work so admirably that he is completely lost sight of; but we must not close without offering him our thanks for the pleasure me have received from Mr. De Launay's work."—*London Mining Journal*, Feb. 27, 1909. (A. R.)

A STUDY OF ORE DEPOSITS FOR THE PRACTICAL MINER, with a Description of Ore Minerals, Rock Minerals and Rocks; a Guide to the Prospector, by J. P. WALLACE, M.D., Member of the American Institute of Mining Engineers, and the Colorado Scientific Society. (Hill Pub'ishing Co., New York.)

"The main ideas which guided the author in the production of this book were to use simple language to make the information concise as well as accurate, and to deal only with essentials, in order that he might supply a thoroughly practical hand-book for the average miner, the prospector, and the mining public. The contents are divided into five parts under the following headings:—Ore Minerals, Rock-forming Minerals, Rocks and Rock Displacements, General Characters and Classes of Ore Deposits, Some Types of Ore Deposits, Mine Valuation and Prospecting. A brief description of the most important minerals is given, and the structural features of ore deposits and the walls enclosing them, with the form, origin, and manner of occurrence of deposits have been treated in some detail. In order to illustrate and enforce his principles, the author has given somewhat full descriptions of many extensively developed mines in various parts of the world. There is a certain raciness in the author's style, which goes a long way to remove it from the neighbourhood of technical books, and which will undoubtedly make this book acceptable to a large circle of readers. The last chapter is devoted to Gleanings from the Mining Field, and topics for discussion around the camp fire, of which the following is a fair sample:—'Every miner and prospector should read. Mining journals and mining books are all important to you. Don't hug to yourself the belief that experience is everything. Open your mind to the reception of knowledge from any and all sources. Practical experience and book-learning combined make the all-round man.'"—*Australian Mining Standard*, Feb. 3, 1909, p. 118. (A. R.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 104/09. Rollo Bowman Ballantine. An improved process for the production of metallic sections, and apparatus therefor. 3.3.09.

(P.) 105/09. Edward Macartney. Improvements in means for distributing tailings or the like on mine dumps, applicable also for other analogous purposes. 4.3 09.

(C.) 106/09. Charles Christiansen. Improvements in valves for rock drills, pneumatic tools and the like. 4.3.09.

(P.) 107/09. Henry James Shedlock Heather. An improved method of protecting electric apparatus against damage from overload. 4 3.09.

(P.) 109/09. George Hans Hirsch. Improvements in tube mill liners. 5.3.09.

(C.) 110/09. Hon. Charles Algernon Parsons, C.B. Improvements in and relating to controlling devices for plant using elastic working fluid. 5.3.09.

(P.) 111/09. David Robertson. Improvements in rock drills and similar pneumatic tools or machines. 6 3.09.

(P.) 112/09. William Arthur Caldecott. Improvements in or connected with dissolving vats for use in the treatment of ores. 6.3.09.

(P.) 113/09. Isaac Stern (1), Frederick Henry Boose (2), trading under the style or firm of I. Stern and Company. Improvements in the manufacture of metallic paints and enamels and the like from mineral ores. 6.3.09.

(P.) 114/09. David Robertson. Improvements in rock drilling machines, reciprocating engines and the like. 9.3.09.

(P.) 115/09. James Chalmers Johnstone. Improvements in pulleys or idlers for conveyor belts, also applicable for other purposes. 9.3.09.

(P.) 117/09. Sidney Joseph Crowther Wilkinson. Improvements in means for securing bottle closures. 11.3.09.

(C.) 118/09. Edwin Burt. Improvements in filters. 12.3.09.

(C.) 119/09. Jan Bastiaan Cloete. A petrifying and waterproof liquid. 13.3.09.

(C.) 120/09. Hans Charles Behr. Improvements connected with rotary ore feeders. 13.3.09.

(C.) 121/09. Arthur Symons. Combination pocket tool. 13.3.09.

(P.) 122/09. William Calder. Improvements in the treatment of ore by wet methods. 13.3.09.

(P.) 123/09. Hans Charles Behr. Improvements in stamp tappets. 13.3.09.

(P.) 124/09. Reginald Crylle Thomas. Improvements in valves for rock drilling and like machines. 15.3.09.

(P.) 125/09. Charles Edward Harris. Improvements relating to stoppers for bottles and the like. 15.3.09.

(P.) 126/09. Tullio Bubola. Improvements in or relating to tanks or vats for treatment of crushed ore products. 16.3.09.

(P.) 127/02. James Stevenson (1), James Falgate (2). A new or improved tool holder applicable for lathes and turning machines. 16.3.09.

(P.) 128/09. Alexander Pearson. An automatic feeder. 16.3 09.

(P.) 130/09. Samuel Raybould. Improvements in stamp guides. 18.3.09.

(C.) 131/09. Hugo Velton. Improvements in conveyors. 19.3.09.

(P.) 132/09. Charles Reeves. Improved method for the recovery of gold and other metals in mills or reduction works. 19.3.09.

(C.) 133/09. John Baptiste Sauve (1), Frank Bernadi (2). Electrical fuse spitter. 19.3.09.

(C.) 134/09. Benjamin Hastings. Improvements in or relating to rock drills. 19.3.09.

(C.) 135/09. Werther Anders Gustaf von Heidenstam (1), Karl Louis Felix Friedmann (2). Process of impregnating wood. 19.3.09.

(P.) 136/09. George Elias George (1), Carl Voss (2). Improved apparatus for the production of motive power. 20.3.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices

ADAMSON, R., *l/o* Riverlea Mine ; Susanna Mine, Gwanda, Rhodesia.
BAWDEN, F. A., *l/o* Redruth ; Knights Deep, Ltd., P. O. Box 143, Germiston.
BARRY, E. A., *l/o* Johannesburg ; Knights Central, Ltd., P. O. Box 91, Germiston.
CALDER, W. L., *l/o* Warmbaths ; P. O. Box 2531, Johannesburg.
DAVIES, E., *to* Yankee Doodle Mine, Selukwe, Rhodesia.
DODDS, J. B. K., *l/o* Johannesburg ; Geduld Proprietary Mines, Ltd., P. O. Box 41, Springs.
HOLLIS, E., *to* P. O. Box 1123, Johannesburg.
GRIFFITHS, A. P., *to* 781-2, Salisbury House, London, E.C.
HUGHES, HUGH, *to* Ventanas Mining and Ex. Co., Ltd., Villa Corona, via Chavarria, Durango, Mexico.
JENNINGS, T. B., *l/o* Germiston ; P. O. Box 1565, Johannesburg.
KRAUSE, H. L., *l/o* Roodepoort ; Geldenhuis Deep, Ltd., P. O. Box 54, Cleveland.
LESLIE, C. D., *l/o* Germiston ; 56, Harley House, Regents Park, London, N.W.
LINDSAY, R., *l/o* Johannesburg ; Heidelberg Roodepoort G. M. Co., Ltd., P. O. Greylingstad.
NEL, P. O., *l/o* Germiston ; Robinson Deep G. M. Co , Ltd., P. O. Box 1488, Johannesburg.
ROSE, A. F., *l/o* Springs ; 748, Warren Street, Hudson, New York, U.S.A.
SMITH, EWING, *l/o* Sarawak ; 4, Queen Street, Craigie, Perth, Scotland.
WILEY, H., *l/o* Ireland ; P. O. Box 111, Salisbury, Rhodesia.
WISDOM, G. E., *l/o* Salisbury ; Yankee Doodle Mine, Selukwe, Rhodesia.
WOODS, W., *to* P. O. Box 185, East Rand.

THE JOURNAL
OF THE
Chemical, Metallurgical and Mining Society
OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. IX. MAY, 1909. No. 11.

Proceedings
AT
Ordinary General Meeting, May 15, 1909.

The Ordinary General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, May 15th, Mr. R G. Bevington (President), in the chair. There were also present:—

46 Members: Dr. J. Moir, Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, K. L. Graham, A. Richardson, C. B. Saner, G. O. Smart, Prof. G. H. Stanley, J. E. Thomas, H. A. White. Prof. J. A. Wilkinson, W. A. Caldecott, W. Cullen, H A. Adams, A. Avent, M. Baumann, S. Beaton, W. Broom, G. L. Burnett, A. A. Coaton, J. B. K. Dodds, L. Evans, J. H. Hamill, M. C. Inglis, J. H. Johnson, G. A. Lawson, H. Lea, C. E. Meyer, H. Meyer, P. T. Morrisby, S. Newton, H. W. Pidsley, S. C. Quin. O. D. Ross, W. Sharp, S. Shlom, A. L. Spoor, R. S. Stokes, J. Strang, C. T. Sydenham, J. A. Taylor, A. Thomas, O. Tonnesen, and L. J. Wilmoth.

14 Associates: Messrs. J. Bowyer. J. Chilton, W. J. N. Dunnachie, J. Hanna, C. Lipschitz, G. H. Little, N. Newland, J. Pope. H. B. Powter, W. S. V. Price, H. Stadler, G. G. Thomas, W. E. Thorpe, and C. Toombs.

25 Visitors, and Fred. Rowland, Secretary.

The minutes of the previous meeting, as printed in the March *Journal*, were confirmed.

NEW MEMBERS.

Messrs. L. J. Wilmoth and W. Broom were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that both the candidates for membership had been unanimously elected, as follows:—

HAILEY, RUPERT CLODE, A.I.M.M., Poleleh, North Celebes, Nederlands India. Mining Engineer.
POTTER, HENRY SAMUEL, 38, Kapteyn Street, Hospital Hill, Johannesburg. Mechanical Engineer and Works Manager.

The President: You will notice from the agenda that one student member was admitted by the Council at its last meeting:—

MUNRO, JOHN STRACHAN, Transvaal University College, P. O. Box 1176, Johannesburg.

GENERAL BUSINESS.

Dr. J. Moir (*Vice-President*): I wish to propose a very hearty vote of thanks to the Dinner Committee. I am sure that we are all agreed that the dinner this year was a great success socially, and I understand, also, that it was more than usually a success financially.

This was carried unanimously.

COLOURED EXPLOSIVES.

Mr. W. Cullen (*Past-President*): At our meeting two months ago you will remember there was a discussion upon the question of coloured wrappers for explosives. I have not noticed that anything has been said on the matter since, but outside the Society a suggestion has been made by more than one individual, that it might meet the point brought forward by Mr. Kotzé (the Government Mining Engineer), if, instead of having the explosives enclosed in coloured wrappers, which was my proposal, we should have the explosives themselves coloured; by that means mine managers and others would be able to ascertain the particular explosive actually used in certain mines, and might even be able to trace accidents to their source. I put it to them that one could hardly get a colouring matter that would not detract from the efficiency of the explosive itself. If the explosive must sacrifice 1% of its strength, would the colouring compensate for this loss of efficiency? It is a small matter, but I think it would be wise for those members who have had practical experience in handling explosives to give their opinion.

Mr. Lewis Evans (*Member*): With regard to Mr. Cullen's remarks, I think one of the main points in suggesting the colouring of explosives was the fact that pieces got into the mortar boxes, and it was thought that if the explosive were coloured, the pieces might more easily be picked out on the sorting tables, whereas a coloured wrapper would get torn off. I myself do not think it is worth while sacrificing any of the power of the explosive for this purpose.

Mr. W. S. V. Price (*Associate*): With regard to colouring the explosive, I am sure cochineal would colour it, and would not make an atom of difference in the efficiency of the explosive. I have been doing some excavating work lately with a certain explosive which I found too strong, and I have been putting tamping in between, sometimes with as little as ½ in. explosive and perhaps a stick of tamping 6 in. or 7 in. long, and firing it with about ½ in. of gelatine. With one explosive I found that to work very well indeed. With another explosive, I have had as much as two and three sticks at the back of the hole and a stick of tamping; firing with a whole stick of the same explosive, which has failed to fire the explosive in the back of the hole. Often when I have been clearing away the dirt from broken ground I have found grey sand which I had used for tamping, which has told me there was gelatine underneath, and so saved me from accidents. Regarding the colouring, I do not think it would make an atom of difference in the efficiency of the explosive. I suggest that both the explosive and the wrapper should be coloured.

Dr. J. Moir (*Vice-President*): I should like to say with regard to colouring with cochineal, that it would do very well as regards tint, but it would not dissolvé in nitro-glycerine Still, there are plenty of other red colours which would mix perfectly with nitro-glycerine, such as the azo-dyes, and such a small quantity would be required, perhaps one part in 2,000, that I think the efficiency of the explosive would not be interfered with. None of the modern sulphonated dyes can be used, as they are insoluble in nitro-glycerine; what is wanted is an unsulphonated red azo-dye.

The President: It appears to me that what may have seemed a small matter to begin with, has grown into a subject of considerable interest, and I would certainly ask all members of the Society who have anything to do with explosives to consider this matter carefully, and try to come to some definite conclusion on the point, as to whether it would be advisable or not, to have coloured explosives, or coloured wrappers, or both. I think their opinions would be of very great value to the Society. For myself, although I do not have the handling of explosives, yet from my experience on these fields, it seems to me that to have some method of distinguishing a particular make of explosive, or of seeing whether there is any explosive material present in the dirt, would be very valuable, and I should certainly like to see the matter gone into very much further, and taken up generally. Of course, as Mr. Kotzé mentioned at one of our previous meetings, it might considerably aid the Government Mining Engineer's Department in determining what explosive was used in the case of a serious accident.

THE PRECIPITATION OF GOLD FROM SOLUTION BY CARBONACEOUS MATTER.

Mr. W. Cullen (*Past-President*): I do not know that I have anything to say on the matter which is the least bit new, but I have been struck, on reading this communication of Mr. Caldecott's, with the importance of a matter which we have all omitted to take into consideration for so many years. I am certain that, be the amount large or small, everyone will examine their lime more closely in the future than in the past. Following up Prof. Wilkinson's remarks, it must have struck us all that the method of burning lime here has hitherto been a bad one, and that the old method of burning lime which obtains in this Colony and in the Orange River Colony must be done away with at a very early date. If lime of proper quality is to be obtained, the only way is to calcine it in kilns of the Hoffman type, or by producer or Mond gas. The difficulty is, that most of the concerns which go in for the making of lime are very small propositions. I suggest that if lime is going to be obained satisfactorily in future, the mines must take the matter up and themselves establish proper works for the burning of lime.

ELECTION OF SCRUTINEERS FOR THE ANNUAL ELECTIONS.

The President: The next business is to elect scrutineers for the annual ballot; a very important matter in our Society.

The following gentlemen were elected:—Messrs. J. Gray, R. Stokes, P. T. Morrisby, L. J. Wilmoth, W. Beaver, T. T. Nichol, S. Newton and A. D. Viney.

Mr. W. Cullen: I would ask the Council to take into consideration the advisability, or otherwise, of conducting our elections on the system of proportional representation. We have lost it in the Draft Act of Union, and it might be a consolation to some of us to be able to conduct our own elections in that way.

The President: I understand that proportional representation has been thrown out by the Union Convention. I do not think I will ask Mr. Cullen to-night to tell us what it really means. As far as adopting the principle is concerned; if you wish to bring it forward, it should be brought before the Council, and then, before a special general meeting.

Mr. W. Cullen: I merely suggest that it may be kept in view.

The President: The matter will be kept in view.

STOPE MEASUREMENTS.

By Otto S. Tonnesen (Member).

The attention of our mining men has long been directed towards a really sound method for the measuring of stopes. Since the question of correct fathomage has from time to time been a point upon which the miner has expressed considerable doubt, it is most desirable for the development of our contract system, and the encouragement of good mining, that some clear and correct method of measurement should be adopted.

The measurement of stopes was one of the main grievances during the late strike, and a Commission was appointed at that time to go into the question. It produced excellent results as regards rules for the contract system, and collected valuable information as to the systems of measuring at present in use, expressing at the same time, however, the opinion that it "did not consider it profitable to discuss in detail the technical merits of the different systems which can be and are employed." The following remarks are extracts from the said Commission's Report :—

"The first task the Commission set itself was to trace the reasons which have led to the grievances which the miners have with regard to the measurement of their work. The evidence which was led on this point was voluminous, and the investigation was conducted in as thorough a manner as possible."

"Your Commission cannot conceal the fact that there seems to be a general suspicion in the mind of the miner that the correct measurements are not given to him. The opinion was repeatedly expressed that the figures of the surveyor must have been tampered with in the Mine Office."

"Many miners, however, candidly admitted that the mine surveyor was not to blame for this, and that he could have no sufficient motive for wilfully giving wrong measurements or in conniving at his figures being altered."

The Commission came to the conclusion that, in order to prevent and minimise disputes, it is essential that every means should be devised whereby confidence in the mine surveyor could be established in the mind of the miner, and sets out certain recommendations in order to promote such a desirable state of affairs. The Commission recommended :—

(1) That the method of measurement be left to the mine management.

(2) That complete written contracts embodying terms be drawn up.

(3) That miners' shall have the right to call in an independent certificated mine surveyor.

(4) That in the event of the measurements of the mine surveyor and contractors' surveyor agreeing within 3%, the mean of the two measurements should be taken ; but in the event of a greater difference, the matter should be referred to the Government Mining Engineer for decision.

(5) That the Government Mining Engineer shall have the right of withdrawing or suspending mine surveyors' certificates after enquiry by the Board of Examiners.

(6) Certain rules for settlement of cost of check survey, etc.

(7) That plans of stope-faces be properly kept and certified.

(8) That such plans be plotted on a scale of 1 to 250 on the plane of the reef.

Some of our best mine surveyors were examined by the Commission and gave valuable evidence. The Mine Surveyors' Association also submitted for the Commission's use an exelent paper, explaining the different methods of measurement. The surveyors examined agreed to a rough estimate of 2% as the average error in the work, no attempt being made, however, to show clearly how the figure was arrived at, nor to criticise both theoretically and practically the different methods in use, in order to define the limits of error, nor, finally, to introduce improvements which would keep these errors within certain allowable limits.

The measures suggested by the Commission are undoubtedly of great importance, but it cannot be denied that the root of the evil lies in the defects of the methods of stope measuring employed. It seems, therefore, that the most effective way of inspiring confidence in the mine surveyor would be to improve his methods of stope measuring, so that the surveyor himself may have full and firm reliance in his own work. At present such is not the case, and he is only too often the victim of accumulated errors and puzzling distortions. Furthermore, the methods in use are sources of hardship and sometimes of danger, which in themselves are often the cause of slips, omissions, and mistakes. It will be the object of this paper to try to improve the present systems of stope measuring, by utilising to its full extent the information available in the office, and by introducing a mathematical and correct method of stope measuring ; this method having also the advantage of being easy, quick, and simple.

The following are some of the main objections raised against the methods of stope measuring at present employed on the Rand.

1. *Objections to Stope Measuring with Hanging Compass and Clinometer.*—Mr. Dunbar D. Scott writes in his paper on "Mine-Surveying Instruments."*

"Compass dials are used, as a rule, by the least informed of our mining men, whereas their appropriate association should be only with expert students of physical science. The compass, as applied to mine surveying, is a relic of our ancient ancestry, and still there are those who cling to it in the knowledge that it is capable of losing its magnetism; that its geometric and magnetic axes may not coincide; that it is susceptible to the influence of local attractions; to diurnal variations, which change with the latitude, temperature and season; to secular variation, which has been as great as 24 degrees; to an annual deviation; to lunar inequalities and solar phenomena; to irregular and intangible magnetic storms; to the horizontal intensity of the earth's magnetism; to electric trolley or lighting systems, which have been known to reverse the poles; that the point of support may become blunted; that it may be set eccentrically, or become bent, so that the opposite ends indicate different readings; that to attach a vernier to such a device is a civilised barbarism; and that the line of collimation in the telescope is rarely coincident, or parallel, with the meridian line of the compass."

Another objection to the method of measurement by the hanging compass is that the inclined distances have to be reduced to horizontal distances, the survey plotted on a horizontal plane, then the calculated area stoped, converted back to the plane of the reef again. This last objection will be further dealt with under the method of measurements by the theodolite.

2. *Objections to Stope Measuring with Theodolite.*—In a steep stope the setting up of a theodolite is a very lengthy and difficult operation. The method requires the observation of a great number of angles and their booking, and, as a surveyor's position on such steep inclines is bound to be trying and uncomfortable—since he must take great care to avoid touching the instrument legs, or otherwise disturbing the adjustments of the theodolite—these may often be a source of errors. The office work is slow, as first the inclined distances must be reduced to the horizontal; then the survey plotted on a horizontal plan and finally the horizontal area converted back to the inclined area.

In all these operations errors may creep in, and this is particularly the case in the last operation, viz., the converting of the horizontal area into the corresponding inclined fathomage. In a stope dipping 60°, an error of 1° in taking the dip affects the fathomage by 3%, and an error of 2° affects the fathomage by 7%. In a stope dipping at 45°, an error of 1° affects the fathomage by 2%, and an error of 2° by 4%. It may perhaps seem an easy matter to determine the average dip on the area stoped accurately, but practice shows that such is not the case. The tendency is to measure the dip too flat. The face of the stope bounding the area mined is irregular and slanting away from the dip line, towards the winze. To obtain the average dip, therefore, is a matter of weighting the dips observed, by the areas bounded by them, and is in practice really a matter of judgment on the part of the observer. Sections drawn up for the determination of dip add further to the uncertainty and errors by the defects of plotting and protractor readings, and, further, this method necessitates the use of a surveying instrument of great accuracy merely to obtain an approximate result. The added possible errors exceed considerably the limit of error suggested and approved by the Mines Regulations Commission, viz., 3%.

3. *Objections to the Tape-Triangulation Method.*—Although this method seems theoretically the best, it nevertheless has many drawbacks in actual practice. Often the old pegs are inaccessible and of little or no use, owing to pillars, stulls, or the steepness of the stope. This last difficulty may, of course, be overcome by using ropes, but as the direction of the stope face is slanting and not on the dip of the reef, several ropes will often have to be used, involving a considerable loss of time. In a long-backed stope the surveyor will often not be able to stretch his tape from top to bottom without the tape touching the hanging or footwall, catching in stulls, pillars, or jumpers, and consequently diverting the line of the tape from that of a straight line. But even when such is possible, sag in the tape is unavoidable, and also long and inaccurate offsets.

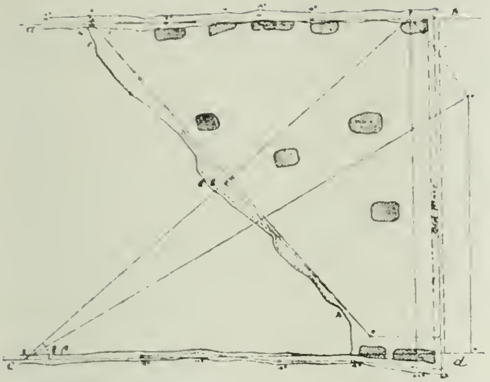
The Sag in the Tape.—Fig. 1 represents a stope plan on the plane of the reef. The backs here were 250 ft. The dip of the reef 50°. A tape was stretched from top to bottom of stope, *i.e.*, the line AB and the working face of the stope measured by offsetting from this line.

In order to determine the effect of a sagging tape on the offsets taken to the stope face, I

---

* Transactions of the Institution of Mining Engineers, vol. xxviii., part 4, p. 649.

FIG. 1.



have drawn a line EF perpendicularly to AB. The tape will hang in the shape of a parabola Ae_1B. Let us, for simplicity, suppose the maximum sag to be at point *e*. Fig. 2 shows to enlarged scale a vertical section through EF. The true dip γ of this line EF is found by construction of a right-angled triangle $EF'K$ (see Fig. 1, where $EF^1 = EF$ and F^1K equals vertical distance between top and bottom drive. The true dip γ of line EF equals angle F^1EK.

FIG. 2.

SECTION E-F

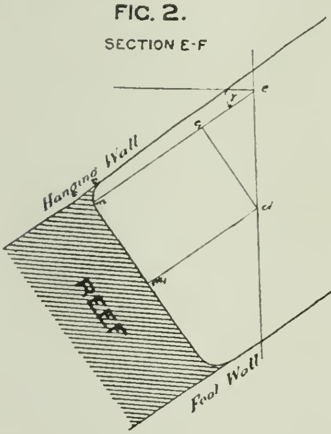


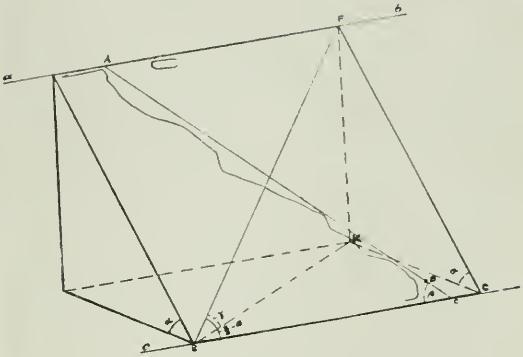
Fig. 3 represents a perspective sketch of the stope, and shows the correctness of the above construction. γ may also be found analytically, as follows (see Fig. 3):—

Let a = dip of reef, and β = incline angle ACE. In triangle FGK we have, (1) $FK = FG \sin a$. In triangle FGE we have, (2) $FG = EF \cos \beta$. In triangle FKE we have, (3) $FK = EF \sin \gamma$, by substitution $FG \sin a = EF \sin \gamma = \frac{FG}{\cos \beta} \sin \gamma$

$$\therefore \sin \gamma = \sin a \cos \beta.$$

Returning to section EF, shown on Fig. 2, point *e* is the mathematical point where the tape would be if there were no sag. Let *e d* be the vertical sag in the tape, then *e m* is the mathematically correct offset, while $d\,m_1$ is the offset as measured underground owing to the sag. Project the point *d* up to e_1 on the plane of the stope. Then distance $e\,e_1$ is the error in offsetting at point *e*. The figure shows that, $e\,e_1 = e\,d \sin \gamma$. That is, the inclined sag $e\,e_1$ is equal to the vertical sag *e d* multiplied by the *sin* of dip γ of section line EF.

FIG. 3.



The following table shows offset errors caused by different values of vertical sag for different dips:—

Dip of Section.	Incline Sag caused by Vertical Sags of					
γ	Feet. 1	Feet. 2	Feet. 3	Feet. 4	Feet. 5	Feet. 6
20°	0·3	0·7	1·0	1·4	1·7	2·1
30°	0·5	1·0	1·5	2·0	2·5	3 0
40°	0·6	1·3	1·9	2·6	3·2	3·9
50°	0·8	1·5	2·3	3·1	3·8	4·6
60°	0·9	1·7	2·6	3 5	4·3	5·2

In Fig. 1 the parabolic line Ae_1B represents the sagging tape as projected on to the stope plane. The full drawn line *f g h* represents the true position of the stope face, while the dotted line $f\,g_1\,h$ shows the false stope face as plotted from the measurements taken underground. The amount of vertical sag is dependent on the length of the line from which the offsets are taken and the condition of the tape. The sag increases as the square of the length of this line. A wet tape increases the sag by reason of its increased weight. The table shows that the inclined sag increases with increased dip γ, *i.e* conversely with the angle β. When β equals 90° the effect of the sag on the offsetting is nil.

In the above example the length of the offset line AB was 316 ft. A maximum vertical sag of 4 ft. will cause an offset error of 2 ft. (see table). Ae_1B is a parabolic segment. The area of a parabolic segment equals two thirds the area of the circumscribed parallelogram. The area of $AeBe_1$ for the above sag will therefore amount to $\frac{2}{3} \times 2 \times 316 = 421$ sq. ft. = 11·7 fathoms. If the monthly area stoped be put at an average of 50 fathoms, the error in such a case would amount to $\frac{11{\cdot}7 \times 100}{50}\% = 23{\cdot}4\%$. The following table shows the corresponding errors for different sags in the above example :—

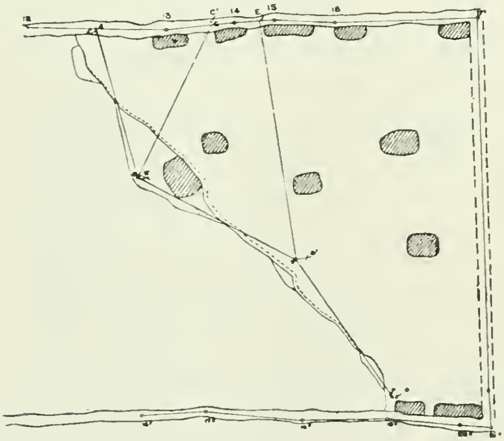
LENGTH OF OFFSET LINE = 316 FT. $\gamma = 30^\circ$.

Vertical Sag. Feet.	Area of parabolic segment $AeBe_1$ Fathoms.	Per cent. error (50 fathom per stope taken as monthly average).
5	14·6	29·2
4	11·7	23·4
3	8·8	17·6
2	5·9	11·8

The true error is, however, greater than shown in the above table owing to the difference in length between the face line and straight line AB.

It may be urged, however, that this sag will occur every time the stope is measured, and the error thus committed only once. Such is not the case, for the sag will be different every month owing to different tapes, different slant of stope face, different length of offset lines, and sometimes it may disappear altogether. The measurement of the next following months stope face in the above example (see Fig. 4) would not be affected by sag to any extent. In order to avoid sag errors some surveyors start all the measurements from the rails; but this practice only replaces the effects of the sag by the far more dangerous errors inherent to this method; for the tape, when held on the footwall, cannot be stretched in a straight line owing to the unevenness of the footwall and to jumpers and protruding pieces of rock, and, furthermore, the rail is not a defined point and may be a couple of feet out from the survey line.

FIG. 4.



The errors, therefore, arising from a sagging tape alone would seem almost sufficient to render this method of stope measuring very unreliable. But there are further sources of error.

The Projection Error.—At A (see Fig. 1) the tape should be held at a point on the straight line joining pegs 12E and 13E (the pegs themselves being only occasionally accessible from the stope). This is, however, impracticable, and in many cases impossible, and is therefore seldom done in practice. The tape is simply held at a mark in the hanging or foot, as the case may be, and the distance to nearest drive peg booked. This mark would, in most cases, project on to the stope plan somewhat above or below the mathematically correct point, for instance at points marked A_1 and A_2 in the above example. Thereby a new error comes into account which I will call the projection error. Figs. 5A and 5B will illustrate this. *a b* is the plane of the reef. A represents the mathematically correct point where the tape should be held, being on the straight line joining two drive pegs. Instead of being held on these points (impossible in the case illustrated on Fig. 5B) the tapes would, for instance, be held at A_1 and A_2 respectively, causing projection errors i_1 and i_2. If the tapes were held at the foot instead of the hanging, a corresponding error would occur.

FIG. 5A

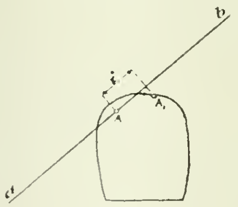
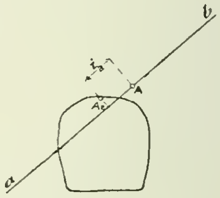


FIG 5B



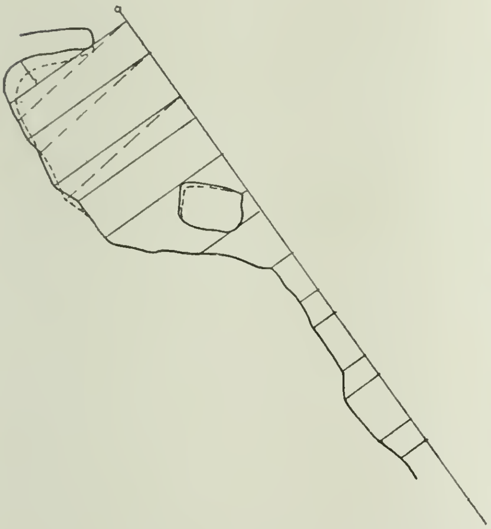
In Fig. 1 the result of projection error would be triangles of error A_1AB and A_2AB respectively. Another result would be wrong intersection at point B.

Intersection Error.—Fig. 4 illustrates a very ordinary case of intersection error. The whole framework for measurement of this stope face is undefined and unreliable, and yet it was the only system which could be adopted owing to the steepness and inaccessibility of the stope. The result of the bad intersections would, in this case, cause an error equal to the area of the polygons A_1BB_1A, BDD_1B_1 and DFF_1D_1.

The elasticity or extensibility of a linen tape will also cause bad intersections. A stretch or slackening of 6 in. merely—which is unavoidable—may, under certain adverse circumstances, cause an intersection error which completely vitiates the work done.

Errors caused by long offsets not being at right angle to the tape.—These errors are not of the same importance as the aforementioned, because the fathomage must necessarily be chiefly dependent on the triangle network or skeleton on which the offsetting is founded. But offsetting errors are nevertheless sometimes of great importance. Fig. 6 shows such cases. The full drawn line represents the true face, the dotted represents the results of offsets askew.

FIG. 6.



It is very difficult, not to say impossible, to judge offsets of from 20 to 30 ft. in length at right angles to the datum line, particularly as the surveyor often, in such cases, has to hang on to a rope to keep his balance. The shortest distance is an uncertain guide when the offset is long and the datum line slack.

The effect of these errors, the projection, intersection and offset errors, cannot be put down into mathematical formulæ because they are so irregular and difficult to trace.

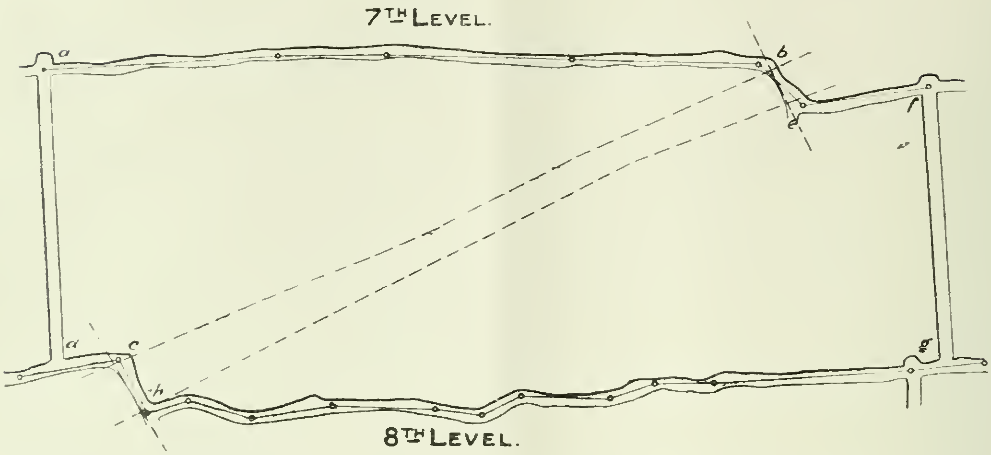
Besides these unavoidable errors there are also other weighty objections to the present methods. They are expensive, slow and complicated. They involve hardships, exhaustion and even danger, which are often the true causes of mistakes and omissions. It may seem, to judge from the plan, an easy thing to choose the best triangle system, but it is quite a different task in practice underground. Ropes may be wanted to gain access to the stope and pegs, and lines that seem easy enough on the plan may be impossible underground owing to inaccessibility, loose rock and dangerous hanging; continuous climbing up and down the stopes is often necessary in order to assure that the tapes are not catching and that they are held on the right place.

I now consider it proved that the present systems of stope measuring involve possible errors far above the allowable error of 3% as proposed for legislation by the Mining Regulations Commission. The members of the latter, together with most mine surveyors are in favour of the tape triangulation method, and it has undoubtedly many good points and practical advantages underground. I have, however, taken more trouble in proving its defects and discussing it in detail, because the method which I now propose to explain is practically built on its basis, meeting its defects, and avoiding its sources of error.

Having thus briefly criticised the most important systems of stope measuring hitherto in use on the Rand, I shall now proceed to explain a new method wherein I utilise as far as practicable, the advantages of the old methods and avoid their sources of error. Before entering into the explanation of this method, I wish to emphasize that in all surveying the object is to arrive at a certain degree of accuracy, and, with this in view, the surveyor's work should be performed as expeditiously as possible.

In practice no stope forms a true plane. We cannot, however, usually for practical reasons, consider any spherical or other curved surface, but are limited to projecting the stopes on to planes. The size and extent of such planes depend on the conditions of individual mines, and it is for the surveyor to judge where and how to break the stope plans off. Figure 7 shows a horizontal plan of a faulted part of a mine. The stopes of this block were plotted and measured in three main sections, viz., *a b c d*, *e f g h* and *b e h c*, due allowance being made within these main sections for curvature of the reef. Cases of this nature are quite common. Not only faults, however, but sudden changes in the dip and strike will often make it necessary to measure the stopes in different sections. My method of stope-measuring employs a survey instrument and tape. By means of these, distances and angles are measured and plotted directly on the inclined plane of the reef. The instrument—which I have termed the "Stereometer"—is so constructed that it will measure the true inclined angle between any three points

FIG. 7



without observing the horizontal and vertical angles, there being no reduction required, and other defects of the afore-mentioned methods are avoided. It is a free application of the principle of the sextant used for astronomical and naval surveys to mine surveying, with certain alterations to suit the requirements of stope-measuring.

Figure 9 shows the latest type of the Stereometer in plan. Figure 10 shows the instrument in section through *a b*. A and B are prisms. Prism B is stationary, and prism A moveable, being firmly fixed to the pivot P. To this pivot is also fixed the top plate or *alidade* L. The top prism, pivot and alidade thus form one solid construction, which is shown separately in Figure 13. The hypothenuse-sides of the prisms are silvered. The prism A is adjustable on the coverplate D (see Figure 10). In Figure 9 A and D are not shown in order to simplify the drawing. Prism B is similarly fixed to the plate E. The inside surfaces of plates D and E are brightly polished so that they serve as reflectors or view finders for the prisms, and thus enable the surveyor to take his sights more rapidly. Plate E is fixed to plate H. H is the limb or bottom-plate of the instrument, and terminates in the handle O. H is shown separately in plan in Figure 15, and Figure 16 is a front elevation of the same. Figure 17 is a side elevation, Figure 18 a section through centre line *g h*, and Figure 19 a section through *e f*. J, shown separately in Figure 14, is a metal plate which is run into the slit in bottom plate H, and, when in position it is clamped rigid by screw K. It is for direct plotting of angles and serves the same purpose as the drawing paper on a plane table. For this purpose the edge of the alidade or top plate is bevelled to enable the angles to be struck off on the plate J. By loosening screw K the plate J is easily detached and replaced by a new one, and a sufficient number of these should be kept in stock. The plates can be applied directly to the plan, and used instead of a protractor. When finished with they can easily be cleaned, and are then again ready for use. In many cases it is convenient to read the angles, and book them in the field-book. To provide for this the bottom-plate is graduated for reading of angles, there being a vernier attached to the top-plate. The thickness of a needle-prick is about $\frac{1}{8}$°, or 8 min., on an 8-in. protractor, and, since it is a waste of time to take closer observations than are absolutely necessary, it is desirable that the stereometer be graduated to 10 min., easily estimated to 5 min. The ordinary traverse necessary for stope measuring is not long enough to accumulate errors of any consequence. Figures 11 and 12 show the arrangement of clamping and slow-motion screw.

From the foregoing description it will be seen that the instrument consists of two main parts, namely (1) the bottom-plate, with lower prism attached, and the handle with clamping arrangement, and (2) the alidade or top-plate, pivot and top prism. The instrument is designed to be held in the hand, and the point N is for centreing under a suspended plumb-bob or plummet-candle.

Adjustments.—(1) The reflecting surfaces of the prisms must be perpendicular to the plane of the bottom-plate. They are properly adjusted by the makers, and are not very liable to be thrown out of position, but the adjustments should occasionally be tested by the observer. To test this perpendicularity of the reflecting

Fig. 9.

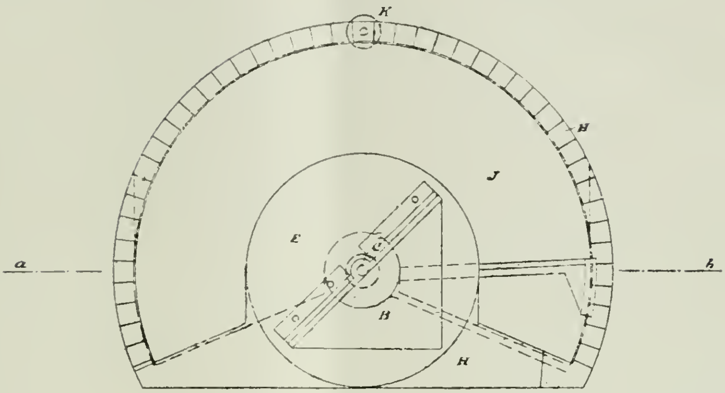


Fig. 10.

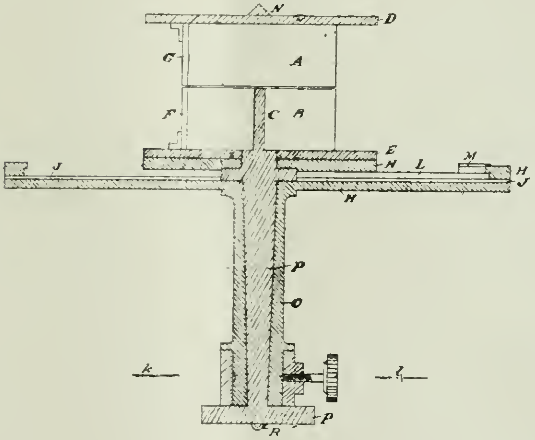


Fig. 11.
Section k-l.

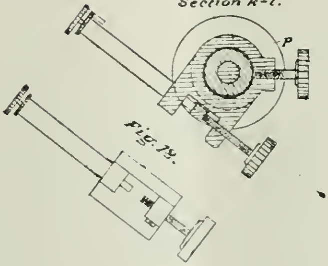
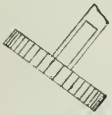


Fig. 12.



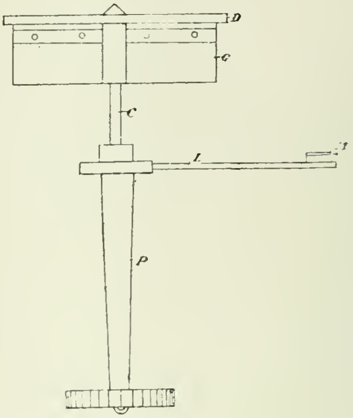


Fig. 13.

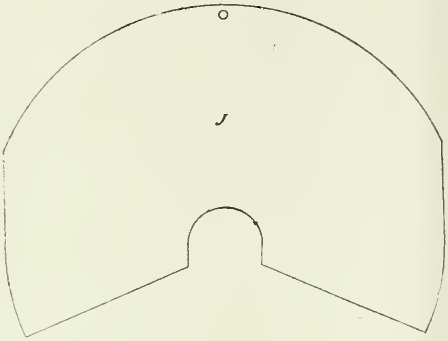


Fig. 14.

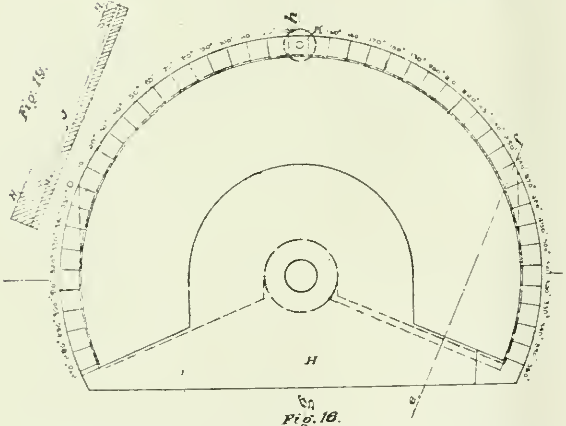


Fig. 15.

Fig. 19.

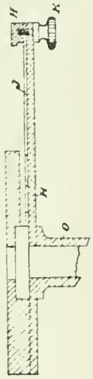


Fig. 18.

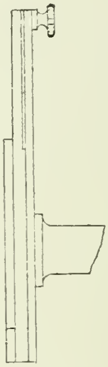


Fig. 17.

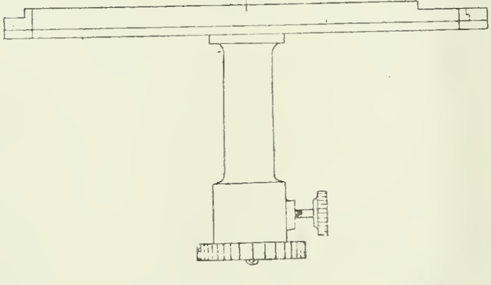


Fig. 16.

surfaces, look into the prisms for the reflected image of the limb arc. The reflected arc, as seen in either prism, should appear to be a true continuation of the arc as seen directly outside the prism. If the prisms are not in adjustment the arc will appear broken at the point where direct and reflected arcs come together. This test should hold good for any position of the alidade.

(2) To determine index errors, mount the instrument on some kind of stand in such a position that the bottom-plate is truly horizontal, by means of spirit-levels. Bring the reflected image of a plumb-line, suspended at some distance, into view in the lower prism. Turn the alidade until the reflected image of the same plumb-line appears in the top prism in such a position as to be a true unbroken continuation of the image in the first prism. At this position the vernier should read zero, and if it does not the reading is the index-error, and a correction should be made to the readings. The error may, however, be corrected by adjusting the vernier. The correction is equal to the error, and is, of course, of contrary sign. If the index stands on

the right of 0°, all the instrument readings are too small, hence the index correction is positive; but if it stands on the left of 0°, the readings are too large, and the correction is negative.

The use of the instrument is, briefly, as follows:—An angle between any three points, "back-sight" point, "set-up" point, and "fore-sight" point, may be found by turning the alidade until images of the "back" and "fore-sight" points coincide at the contact of the two prisms. The angle between the prisms is then equal to half of the angle between the points. The lower plate, however, is so graduated that the correct angle is read directly, 1° of arc being marked as 2°, and so on.

Example: See Figure 20.—Let R and L be "back" and "fore-sight" points, and K the "set-up" point. Let the observer's eye be placed somewhere at O. A candle or plumb bob at R is then reflected to O in the immoveable prism on the bottom plate, while the fore-sight point L is in a similar way also reflected to O through the moveable alidade prism. The angle LKR then equals twice the angle between the reflecting sides of the prims. The position O of the eye is most convenient for observing angles about 180°, because in this position the observer's head cannot come in the way, and also because a symmetrical position of the prisms is ideal according to the laws of refraction. When angles of, say, 90° are wanted, the prisms will have the position shown in Figure 21. LO will then be parallel to the hypothenuse of the moveable prism. In such a case it may be easier and quicker to find the sights by holding the eye at O (in Figure 22) and to take the back sight straight through the alidade prism.

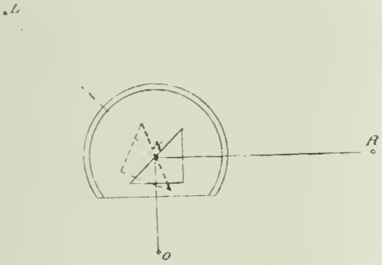


Fig. 20.

The instrument, may, of course, be furnished with a distinct diopter or telescope, but for stope-measuring purposes I find it more expedient to indicate the place of sight, approximately only, by a notch or mark, and to "snap" the angle as soon as the two images coincide, anywhere at the contact of the prisms. A diopter or telescope would naturally reduce the field, and make the observations more difficult. It may at first seem difficult to get the points into view. It is therefore a good plan to accustom one's self to the instrument on the surface before using it underground, in order to become familiar with its movements. It should not be handled *too close to the eye*, but some distance away, strained observations being quite unnecessary. The main point is to hold the plane of the instrument in the plane of the three points observed. Before taking an observation, it is a good plan to make a rough guess of the angle, and set the alidade to it, so that there is little adjusting necessary when taking the sight.

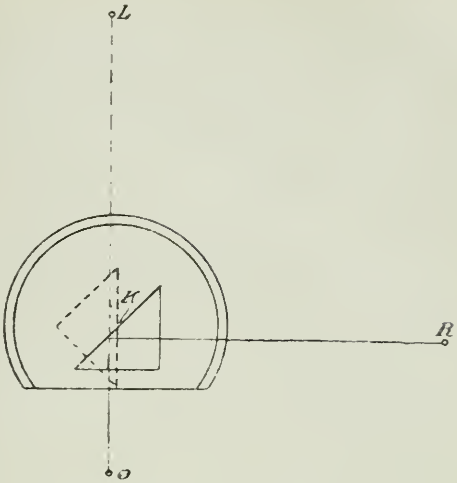


Fig. 21.

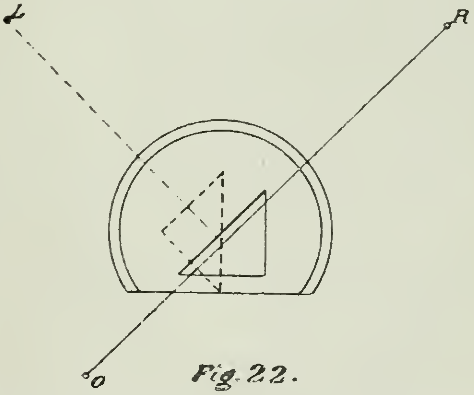


Fig. 22.

When Stope Measuring.—Choose the points and suspend the plumb bobs. Hold the instrument directly under the "set-up" plumb-bob in such a position that the plane of the instrument is approximately parallel to the plane of the stope. It should be so held that the "back-sight" point is in a direction "eyes right" its image being then observed in the lower prism. Turn the alidade until the "fore sight" point appears in the top prism, and, finally, adjust by slow motion screw until the two images coincide. N.B.—The instrument must all the time be held approximately on the plane of the reef, otherwise the images will not appear in the prisms.

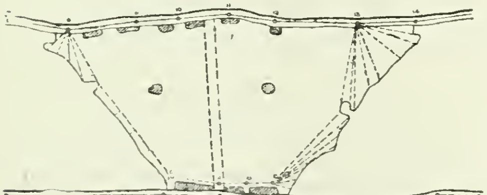
As regards the tape used, it is found more convenient to use one graduated on both sides to feet only, as this is easier to read, distances less than a foot being easily estimated, and by its use the possibility of mistaking feet for links is avoided.

The stopes are measured with this instrument either by radiation on the inclined plane of the stope, by traversing and off-setting directly on the inclined plane, or by combination of both principles.

The Radiation Method.—As a rule the radiation method will prove most expeditious, and most suitable for *flat stopes* where the pegs in the stope are easily accessible and can, therefore, be used over and over again for several months. The principle of radiation is also suitable for *steep stopes* with few pillars, where one "set-up" in the top drive and another at the bottom of the stope is sufficient to overlook the entire stope-face.

Figure 24 represents a steep stope, illustrates the radiation method applied to stopes. The instrument here is set up at peg 13 in the top level, and at point C at the bottom of the stope, and the stope measured by radiating lines, this being done almost as fast as the assistant can climb. Peg C must, of course, be located from the old pegs A and B, or to pegs in the bottom level through boxhole.

FIG. 24.

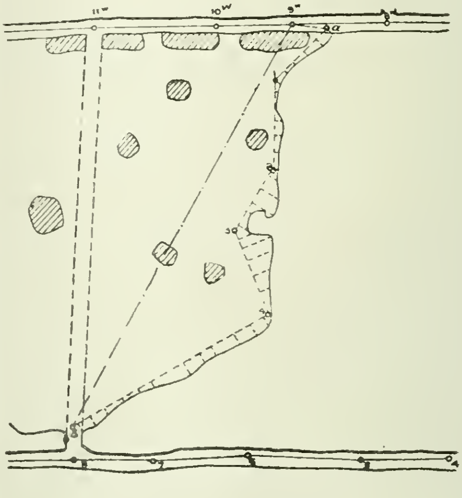


The Traverse Method.—In steep stopes, filled with stulls and pillars, the whole face is seldom visible from the top drive and bottom peg, and intermediate stations will, therefore, have to be put in. To put in a peg hole in a steep stope is, however, trying and very slow work, owing to the awkward positions the hammer boys are compelled to work in. Instead of giving heavy blows they only manage to tap the drills, and at this rate even small holes will take a considerable time. For these intermediate points, plaster of Paris pegs are suitable. They are quickly fixed, and are sufficiently strong to carry a plummet candle. I have found an ordinary small smoke mark to answer the purpose of an intermediate station quite well; but it is more accurate to use plumb-bobs suspended from a leather washer smeared with thick wagon grease, this causing it to stick to the hanging sufficiently long for the purpose, a tin of grease being the only material necessary.

The points sighted to must be in the same inclined plane as that in which the stope is measured and plotted. I have generally found the best position for such sighting points to be at a distance of, say, two or three feet vertically below the pegs: this, however, varies somewhat according to the width of the stope. Points on such a plane are on the reef plane, and can be seen at the longest possible distance. The sighting point may conveniently be a candle held behind a plumb-bob, or, better still, suspended on a line, thus hanging directly under the peg. It is an advantage to use coloured sighting points in order to distinguish from other points in the stope. I have designed a plummet candle for this purpose, with a collar made of coloured mica.

Figure 25 represents a very steep stope full of pillars and stulls. The traverse 9^W, *a*, 1, 2, 3, 4 *b*, was run by measuring inclined angles and distances. Owing to the steepness of the stope the Stereometer was held in the hand. The method of measuring is as follows:—

FIG. 25.



The surveyor plumbs his instrument under peg *a*, the desired vertical distance. The assistant climbs down the stope and fixes a smoke mark or temporary peg at a convenient place say at 1. The surveyor sights and records the angle at 9 *a* 1, and reads the distances 9 *a* and *a* 1. Boys being stationed at *a* and 1 holding the tape, the surveyor takes offsets with an offsetting rod to the stope face from line *a* 1 as he descends the stope. In the meantime, the assistant climbs further down and selects the next station or point, and thus the surveyor continues to survey until point *b* is reached. Point *b*, if not an old peg previously determined, must be located on the stope plan either by connecting down to pegs in bottom drive or by continuing the stope traverse on the west side of the winze up to the top drive, *i.e.* up the opposite face.

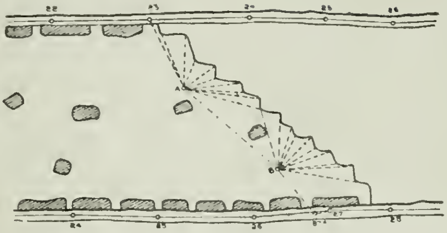
Plotting—The traverse 9^W,*a*,1,2,3,4,*b* is plotted on a sheet of transparent paper directly from the inclined observations. Points 9^W and *b* are joined by a straight line and the tracing paper placed on the stope plan. The points *a*,1,2,3,4 are then transferred on to the stope plan by pricking through and the stope face plotted from the offset measurements.

The effect of the above measurements is the same as if a mathematically straight line $9^W b$ had been available and the stope plotted from the same ; but the traverse being composed of short lines it is not to any extent influenced by errors due to sagging of the tape or long and inaccurate offsets. Owing to the stretching or contracting of tapes, or uneven dip of the reef, the straight line 9^W *b* on the tracing paper may be slightly longer or shorter than the corresponding line on the stope plan. Such difference should be adjusted evenly at the top and bottom peg, but this is of little importance as regards the accuracy of the area stoped, merely tending to lengthen or shorten the stope somewhat and not affecting the fathomage appreciably. Great care should be taken in fixing the top and bottom pegs as the stope advances. The stope shown in Fig. 25 would, without the aid of the stereometer, require several intermediate points to be fixed by tape intersections or by theodolite, and this, owing to the number of stulls and to the steepness of the stope, would take a long time and be liable to considerable errors.

In some cases of stope measuring a combination of the radiation traverse method will be most suitable (see Figs. 26 and 24).

These few examples will illustrate the advantages of the stereometer. Besides its accuracy and simplicity, the following points may be claimed for it :—

FIG. 26.



(1) It affords a continuous and clear method of procedure unobtainable by any other system of stope measuring, and in addition, the work is straight forward and repeated climbing unnecessary.

(2) The risk and difficulty in setting up a theodolite on an incline is obviated by this new method, and nearly all of the hardships met with in stope measuring are dispensed with.

It also combines the advantages of all the old methods, avoiding at the same time their difficulties and sources of error.

It has the great advantage that errors are not accumulated, this accumulation having already proved to be so fatal to the triangulation method under unfavourable conditions. Any two measurements of the same stope, since they are tied up to the drive pegs at top and bottom, must therefore necessarily agree. Thus checks are easily established, and nothing but an actual mistake can cause any appreciable inconsistency in fathomage. Even if some days have elapsed since the first measurement was taken, the untouched benches of the stope will prove whether the first measurement was correct or not, and the data in hand will enable a correction to be made. A mistake in the main traverse, however, is not likely to be undetected when plotting the first survey, as this method gives no room for any "jamming in," which is often necessary in triangulation.

Frequently pegs are blasted out during stoping, and new ones have to be surveyed in. The setting up of a theodolite is slow, and its accuracy is not necessary for this work, and added to this great inconvenience is caused by tramming and other operations, which are carried on in the drive. These points are, however, conveniently surveyed in by use of the stereometer and with remarkable speed. With this instrument the inclined angles of the drive may be observed directly, but in many cases one cannot see down the stope from the drive owing to bad driving, stulls, pillars, etc., and consequently the instrument cannot be held in the true plane of the reef. In this case the new drive pegs must be surveyed in on the horizontal plane and the horizontal angles afterwards corrected for the dip of the reef when plotting on the stope plan (see tables).

Stope Plans.—The filling in of topographical charts in geodetic survey, often done by measuring-table, is always built up and framed on accurate triangulations by theodolite, this being the principle in all survey operations, *i.e.*, to work from the whole to the part, and from the greater to the less. A similar proceeding should be followed underground, where every stope measurement must be based on a triangle-net, which is built up on the stope-plan by joining convenient points to the traverse in the top and bottom drives bounding the stope. The construction of accurate stope-plans is, therefore, of great importance, being the most effective means of detecting mistakes or slips in measuring.

Every block of ground should be plotted separately in order to avoid distortions from surrounding areas, and the stope unfolded, so that the closest approximation possible to the true stope-surface is reached.

The true distance d between any two pegs is found directly from their actual co-ordinates x, y and z (z representing levels). Thus:— $d = \sqrt{(x_1 - x_2)^2 + (y_1 - y_2)^2 + (z_1 - z_2)^2}$ where x_1, y_1, z_1, and x_2, y_2, z_2, represent the co-ordinates of the two points respectively.

(1) Then let
$$x_1 - x_2 = \triangle x$$
$$y_1 - y_2 = \triangle y$$
$$z_1 - z_2 = \triangle z$$

The distance d is now found by a simple and quick graphical construction, as follows (see Fig. 27):—

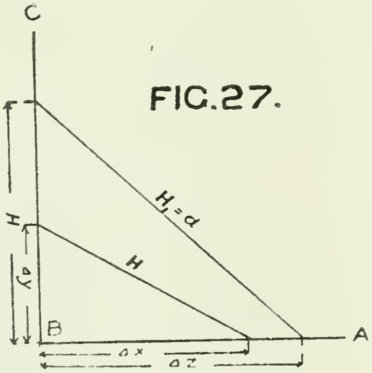


FIG. 27.

Draw two lines, AB, BC, at right angles. Along AB mark off the length of $\triangle x$ to the scale required for the stope plan. Along BC mark off $\triangle y$. The hypotenuse H will then be:—

(2) $H = \sqrt{(\triangle x)^2 + (\triangle y)^2}$. Take H in a pair of beam compasses and mark it off along one line, say, BC. Now mark off $\triangle z$ along the other line AB. The hypotenuse H_1 will then be:—

(3) $H_1 = \sqrt{H^2 + (\triangle z)^2}$ by substituting (2) in (3) $H_1 = \sqrt{(\triangle x)^2 + (\triangle y)^2 + (\triangle z)^2}$ and by substituting equations (1) into (3) we get:—

$$H_1 = \sqrt{(x_1 - x_2)^2 + (y_1 - y_2)^2 + (z_1 - z_2)^2}$$

equals the true distance between the two points in question.

To save the time of scaling off on to the paper, a simple mechanical device may be easily made to do this work by clamping two scales at right angles so that the zero points coincide.

Well conditioned triangles are built up from such distances, intermediate pegs filled in, and a sufficient number of check lines taken. The check lines will give a clear idea of the accuracy of the plan. Many pegs are not on the reef body itself but some distance from it, owing to bad driving or faulting, and these pegs should be omitted in the main triangle net. Intermediate points may be filled in from a table calculated for this purpose.

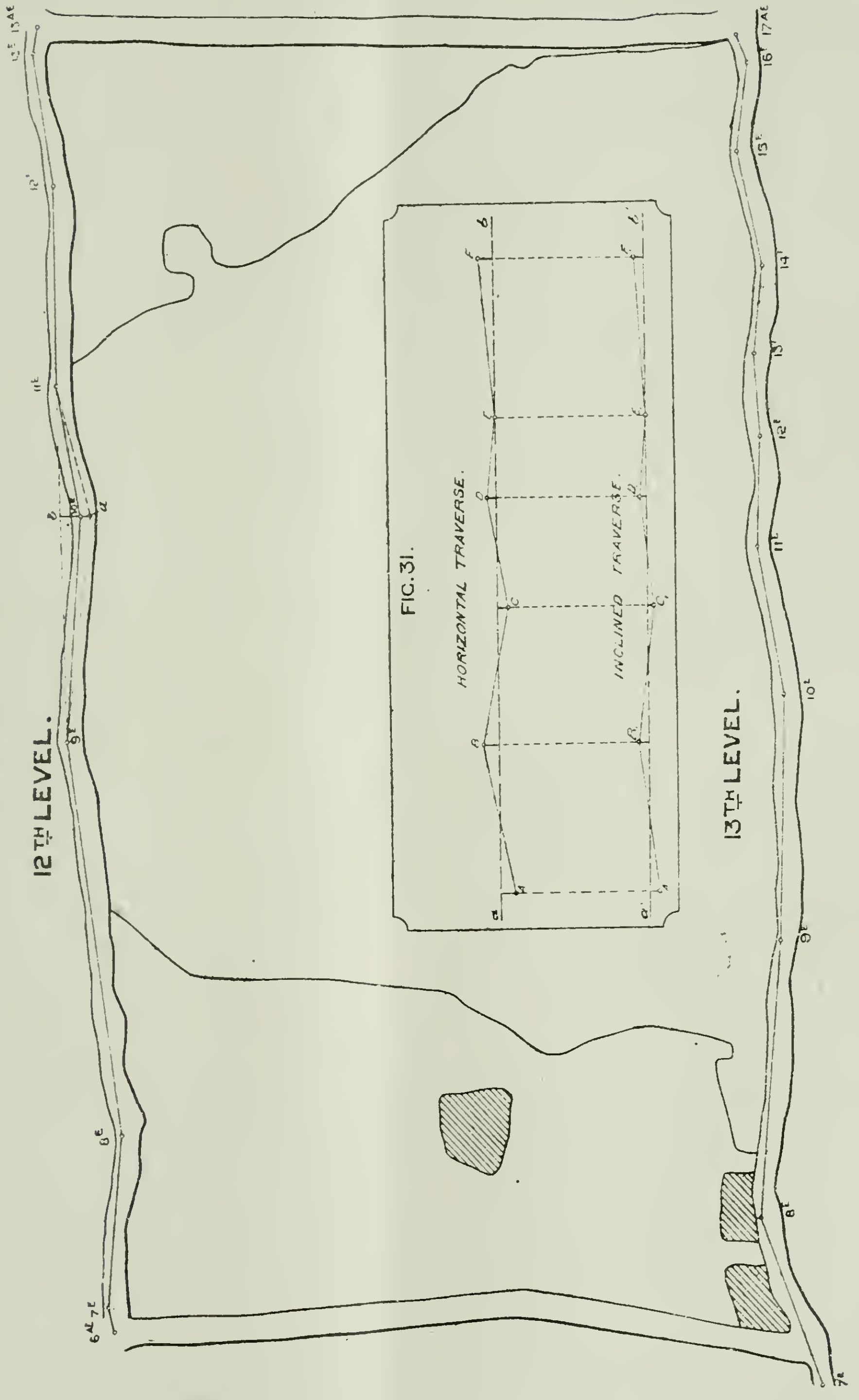
The following example taken from the Angelo Section, E.R.P. Mines, Ltd., will further demonstrate the system. Fig. 30 shows the stope plan of a stretch of ground between the 12th and 13th levels Pegs 8, 10 and 12 on the 12th level, and pegs 10 and 14 on the 13th level, are not included in the main triangle net, since they represent rather irregular driving, while 12, 13 and 16 are excluded from the main triangle net in order to simplify the construction.

The method of plotting this stope plan was as follows:—(See Fig. 30.) The line $9_{12}11_{12}$ was drawn parallel to the top edge of the paper, then point 11_{13} was located by intersection of lines $9_{12}11_{13}$ and $11_{12}11_{13}$, whilst points 15_{13} and 13_{12} were located in the same manner, and so on eastwards up to the winze. In a similar way the triangles to the left of the base line were completed. The intermediate points, *e.g.*, peg 8_{12}, were located by intersections of three lines, here $6^A{}_{12}8_{12}$, $9_{12}8_{12}$ and $8_{12}9_{13}$. The degree of neatness of the intersection gives the necessary warning of any discrepancy, and the centre of the small triangle of error locates the position of the point. All intermediate points were then filled in and the traverse points joined. The winzes were then plotted from actual measurements. To get an idea of the correctness of the plan the lines $6^A{}_{12}10_{13}$, $10_{13}12_{12}$ and $10_{12}8_{13}$ were tried as shown on the plan where the greatest divergence from the true distance was 6 in.

When drawing in the sides of the drive on the stope plan, Table I. converting the horizontal offsets into corresponding inclined offsets, may be used Table I. is calculated for a dip of about 45°, and gives a quick and handy approximation, saving projections or inclined offsetting under ground. In exceptional cases of bad driving the table will not be correct.

This method saves the construction of horizontal plans and vertical sections to a large scale (say, $\frac{1}{250}$). The plan represents an actual unfolding

FIG. 30.


12TH LEVEL.
13TH LEVEL.
FIG. 31.
HORIZONTAL TRAVERSE.
INCLINED TRAVERSE.
a
b
A
B
C
D
E
F
a'
b'
A,
B,
C,
D,
E,
F,
6AE
7E
8E
9E
11E
12E
13E
13AE
8E
9E
10E
11E
12E
13E
14E
15E
16E
17AE
7E

of the stope, and is, therefore, accurate within any reasonable error and free from distortions of any consequence.

Method of Filling in Intermediate Pegs.—In Fig. 30, 9^E and 11^E are pegs fixed on the stope plan. Join 9^E and 11^E. From these two points draw lines which intersect at the point *a*, so that $9^E a 11^E$ make the horizontal angle between these three points. From *a* drop a perpendicular to point *b* and measure its length. Substitute this distance by the inclined distance found in a table calculated for this purpose for the dip at this point. Plot this distance on the perpendicular from its intersection point *b* and call this point 10^E, join and $9^E 10^E 11^E$ will then be the true inclined angle.

This method of transferring horizontal angles to inclined angles leads to a very quick approximation for constructing stope plans. The horizontal traverse A to F (see Fig. 31) of the top drive of a stope is plotted by protractor or co-ordinates, the line *ab* showing the average strike of the reef for the block in question. The inclined strike line a^1b^1 is drawn parallel to *ab*. Project the pegs of the horizontal traverse on to both strike lines. Measure the distances from horizontal pegs to line *ab*, and find the corresponding inclined distance from the table (dips being estimated from plans in $\frac{1}{500}$). From the inclined strike line a^1b^1 offset these inclined distances and join the offset points A_1 to F_1, thus representing the inclined traverse. The inclined position of at least two points of the bottom drive having been located by the co-ordinate method as previously described, the intermediate bottom pegs are filled in from a new strike line as above, or from the bottom stope. This approximation, being a projection, should only be used when the drives on reef are fairly straight.

TABLE I.
On *m* Plane.

Horizontal Offsets.	Incline Offsets.	
	Bottom.	Top.
0	2·	0·
1	2·5	0·5
2	3·4	1·
3	4·	2·
4	5·	2·5
5	5·5	3·
6	6·	4·
7	7·	4·5
8	8·	5·
9	8·5	”
10	9·	”
11	”	”
12	”	”

In a paper of this description it is impossible to include all proofs and references to refraction, etc., and it will be found that in parts abbreviations have been necessary. If, however, any members would like to go further into the subject, I shall be very pleased to give them any information they may wish to have.

During the last six months I have been able to carry out many experimental surveys underground, and these have given ample proof of the accuracy of this method, and its superiority over those already in use. The present form of the stereometer is the result of these investigations, which have shown the defects in my first constructions, and have also indicated desirable improvements.

In conclusion, I wish to thank the Joint General Managers of the East Rand Proprietary Mines, Messrs. J. Morris and S. Penlerick, for their valuable assistance, and the facilities for carrying out this work with which they have provided me; without this the work could not have reached its present stage. I am also greatly indebted to Mr. C. Horst, before whom I first laid my ideas, and whose advice and criticisms have been most valuable. Further, I have to thank Mr. W. L. White, the Underground Manager of the Angelo Gold Mine, where these experiments were carried out, and also Mr. J. Whitehouse for advice in drawing up this paper.

The President: I wish to propose a very hearty vote of thanks to Mr. Tonnesen for his most interesting paper, which I am sure will prove of great interest to our members connected with mining and surveying. The matter of stope measurements is a great thorn in the side of surveyors, mine managers and mine captains. I should also like to congratulate Mr. Tonnesen on the ingenuity he has shown in devising his instrument, which I am sure will be of great use in rapid methods of measuring and surveying stopes.

Mr. A. Richardson (*Member of Council*): I have much pleasure in seconding the vote of thanks to Mr. Tonnesen for the able way in which he has dealt with a subject of the greatest importance both to the surveyors and the mining industry generally, and for the ingenuity he has displayed in the invention of an instrument which promises to considerably lighten the labours of surveyors on "measuring up" days. As I intend to test the instrument I will reserve my remarks on it and on the paper for a future occasion.

Mr. C. B. Saner (*Member of Council*): I should like to support the vote of thanks; I was a mine surveyor some years ago, and frequently had to hide myself for two or three days after the end of the month. The poor unfortunate mine surveyor was always supposed to be "doing the miner down," although he was doing his best. I hope that those of our members who are mine surveyors, samplers and miners will criticise this paper, and bring their views, experiences, and ideas forward. I think the author well deserves the thanks of miners and surveyors.

Mr. W. S. V. Price (*Associate*): The last speaker pointed out that the surveyor has generally to make himself pretty scarce after "measuring up." This paper brought me to Johannesburg to-night, as I hoped to discover the reason for the discrepancy between the miners' measurements and the surveyors' measurements. We all know the miner who drills 5 ft. holes, finds 3 ft. sockets, and yet reckons he has broken a fathom of ground with four holes; but there are others who do break the ground, good miners, and good workers, but who do not get paid for the ground they break. There are some things which, to a certain extent, have been explained to me this evening. One is this sagging in the tape and long distances between pegs. But that is easily remedied. I think we should use steel tapes and shorter lines, and if the boys cannot do the work you must have more white men. With regard to the errors pointed out here, errors in reading, and the difficulties of taking a clear observation, the same may apply to the sextant. Before I came out to this country I was following the sea for many years, and made my way about the ocean with the use of a sextant, but it has to be read rather closer than ten minutes (of arc). If you take a lunar observation ten minutes out you might find yourself about fifty miles inland. The simpler your instrument the better. If you devise an instrument that will simplify the measurements the less likelihood there is of mistakes, for we all know that multiplicity of figures leads to error. With regard to the alteration of stope plans from one plane to another, I cannot see why the stope plan should not be kept on the plane of the reef and, if necessary, owing to change of dip, make it in two sections. We know the surveyors have an awful lot of work to do in a very short time. As to danger and inconvenience of steep stopes, bad hanging, etc., the miner has to put up with all that, and sometimes with a machine hanging round his neck. Whenever I was underground, and the surveyor was measuring up, I was always willing to help him. Mr. Tonnesen's stereometer certainly seems very ingenious, and some such thing ought to make mine surveying very much simpler. I have, therefore, very much pleasure in supporting the vote of thanks to Mr. Tonnesen for his paper and his invention, which I am sure will be a great help to everyone connected with mining.

The vote of thanks was cordially agreed to.

Mr. O. S. Tonnesen (*Member*): I thank you for the way you have received my paper, and for your kind remarks.

Mr. Price claims that it is of no consequence whether the boys have to do repeated climbing. To this I agree, except for the waste of time, but I referred to the surveyor. It is desirable that he should be spared any unnecessary climbing so that he may lend his attention to the more important part of his work. Mr. Price suggests that the surveyor should ask the miner to give him assistance in holding and adjusting tapes. This might in some cases be adopted with advantage if you know your man, but would be a dangerous practice.

As to comparing my instrument to the sextant, I wish to point out that the sextant is too slow in use, owing to its telescope and limited field of view Its excentricity and limitation to small angles are also great objections.

Mr. Price supports my views of using shorter lines in order to avoid sags, but he recommends the use of steel tapes. I am not in favour of steel tapes for stope measuring, because they are cumbersome in practice and rather increase than decrease the amount of sag owing to their greater weight.

The President: The hour is getting late, and it was resolved at our last Council meeting that somewhere about the hour of 9 o'clock discussion should be commenced, because the papers have been numerous and of varying interest, and we generally found that it is nearly time to close the meeting before there was any time to bring on discussion; therefore I propose to act on the resolution of the Council, and postpone the reading of the next two papers; and to take the discussion on the several papers on the agenda.

WHITE LABOUR IN MINING.

(*Read at January Meeting, 1909.*)

By Tom Johnson (Member).

DISCUSSION.

Mr. W. S. V. Price (*Associate*): It may seem a little extraordinary for one who is practically a novice in mining matters, to criticise so experienced a miner as Mr. Johnson, but there are one or two things that struck me on reading his paper in the *Journal*, and so far as I know they have not been touched upon. I was pleased to see that in one case the author mentions that a man running three machines would not do 50% more work than if he were running two. On paper, the theory is that if you got 50% more machines you broke 50% more ground, but like many other theories, it does not always work out, for a man cannot be in three places at once. That is the main trouble. There are some who say that a white man cannot shovel as much rock as a native. The author says in his paper that if he could not do the same work and better than a nigger he would like to change his colour. Just before the first parliamentary elections in this Colony* my old mother wrote and asked me, "how was it there were so many white men on the streets, whilst the mines were wanting labour?" She asked, "could they not do the work, or were they too lazy?" My idea in regard to this being a white man's country, is that the white man should be the boss all the time, not that the work should degrade a white man—far from it; I have no fancy ideas of that kind; I believe, with Mr. Lionel Phillips, that when a white man does that kind of work he should prove himself the better man. When I went to learn mining, I started on the Village Main Reef mine. I was on the machines, but at one time machines were not available, and the shift boss asked me if I would mind going lashing. I averaged 15s a day tramming, but I worked for it. If they could get Kafirs to do the work at 3d. a truck, why should they pay a white man 1s. for the same work; for I never heard anyone except "Labour Members" suggest that mines were to be run as benevolent institutions. My objection to the unskilled white man underground is, that it is putting him in competition with the Kafir. You have had living proof of what white unskilled labour does underground, how it lowers wages, etc. During the strike the mines were flooded with Dutchmen and others, at from 5s. to 10s. a day, and it was certainly against the miners interest, for I know of some even now who are running three and even four machines for 16s. 8d. a shift. As it is now, there is very keen competition, even amongst skilled men. Of course there is always room at the top, but you have to get there first. One manager told me that they had 90 men, lashers, etc., pass through their hands in thirteen days. As soon as they got a few shillings they gave it up. As the author points out, their hearts were broken; they were probably mostly clerks, men who had never done a day's hard work, *i.e.*, manual labour, in their lives. On the Village Deep some years ago, out of their staff of lashers underground they had one bank manager, a ship master, and a tea plantation manager. These fellows, however, had grit and backbone, and stuck to their work; in one case I saw a man's hand quite raw. The men the author has referred to went there just to get a day's pay. Then he goes on with his figures as to pay, etc., under certain conditions. Well, you can make figures prove pretty nearly anything you like. He speaks about men running ten and twelve machines, but a man cannot be in three or four places at the same time, and although he may order a hole to be run in a certain direction, as soon as he leaves the machine, the chances are the holes go in some other direction. He mentions young lads going down, say, as drill collectors, young white lads instead of Kafirs. Now a Kafir will go underground for six or twelve months, then he buys a couple of wives, and he is a made man, and his mining days are over, unless drought or locusts send him back again. Suppose you put your son in at that work. That boy eats just as much dust as a miner. A man who contemplates sending his children underground like that is not going to do it unless they are starving. The average life of a miner is about seven years. Are you going to kill your boys before they are twenty? A youngster should become big and strong before he goes underground, under the conditions obtaining to day. What inducement is there for a youngster to go underground? There are some mines well ventilated, but in the greater number of the mines on the Rand, the ventilation and sanitation are in a very bad condition. I said before the strike, and have said it since, "every man knows that wages must come down, and things find their level some day," but in the meantime the owners will have to spend a lot of money on ventilation, etc. If a man's life were made safer underground he would be able to work there for lower wages, for if a man's working life is twenty years, he does not need to earn as much in the first five years, as he would if five years represented his working life; but at the present time it is no use sending your boys underground, unless you have got them well insured; although it would certainly solve the question of "what to do with our boys!" Regarding surveyors, there is amongst mining men a very black name given to one particular group of mines—"no names no pack drill." I worked on one mine where the manager asked me, "would you recognise your working face if you saw it on the stope plan?" As soon as I saw it I knew I could draw a better plan with my eyes shut. It was absolute robbery. Where I had a 12 ft. or 14 ft. box hole, they had just put a 6 ft. box hole in the plan. The drive pillars were 16 ft. thick in the plan instead of about 6 ft. thick. They gave us £3 for each box hole it is true, but from box hole to box hole you had to cut away 10 ft. of ground for nothing. That is a great inducement for a man to send his children underground, is it not?

---

* 1907.

SOUTH AFRICAN COALS AND THEIR ECONOMICS.

(*Read at April Meeting, 1909.*)

By Arthur J. Andrews.

DISCUSSION.

Mr. W. Cullen (*Past-President*): I think this paper merits a good deal of discussion, and taken in conjunction with the remarks made by Mr. McArthur Johnston, it is, in my opinion, one of the best papers we have had before the Society for some time. One of the first things which struck me when I came to this country was the curious black colour of the coal ash dumps which one saw in the vicinity of every boiler plant. I naturally looked into the matter, and found carbon in astonishing quantities. Recently these same dumps have quite changed their colour, a result of more careful supervision of the stoking, but still there is too much carbon. However, investigation and reflection showed me that it was impossible at this altitude, and with the comparative "thinness" of the air, to expect combustion to go on quite as satisfactorily as at the coast. In any case, the author's explanation for the comparative failure of some coals, confirms what I have found over and over again. The calorific value is far from being everything, and buying coal having a very high ash on a purely calorific basis, is more often than not merely throwing money away. With one coal, which I recently tried, the stokers solemnly assured me that for every five truck loads of coal which they threw on the fire, they had to clean out six trucks of ashes. In any case, though this coal was very cheap, as a result of its low calorific value for steam raising purposes, clay would have done almost as well. There is one point which the author has not referred to, but which is of very great economic importance, viz., the effect of "weathering" upon coal. Not only are the burning qualities affected, but the coal loses greatly in weight as all know who have much to do with the question. I would suggest that this is a subject worth looking into by some of the chemists.

I now pass on to Mr. McArthur Johnston's remarks, to which, however, I have already referred. Among other things, he points out that steam would die hard as opposed to electricity, if greater care and attention were paid to efficiency. I might cite two parallel cases, viz, coal gas, and the old "Chamber" process for the manufacture of sulphuric acid. Theoretically, both of these processes were dead years ago, but in practice coal gas is far from being dead—indeed, it is in a very flourishing condition, and more sulphuric acid is being made to-day by the old "Chamber" process, than ever there was before. The explanation you all know, but it is wonderful all the same.

Reference was made by the author to the necessity for choosing the proper coal for certain conditions, but it must be remembered that few mine managers are their own free agents with regard to the choice of coal, consequently very great anomalies exist on certain mines, coal being frequently used for boiler installations for which it is unsuited in many cases, a large proportion of the heat going straight up the chimney. This is a very important matter indeed, and one which does not receive that attention which it merits.

I have already referred to the question of stoking, and at an early date hope to place the records of an instrument which records the percentage of carbonic acid in flue gases before you. So far it has not been worked long enough to entitle one to say that is an unqualified success, but everything points to its being a very thorough check on the work of stokers.

Mr. A. McA. Johnston (*Vice-President*): The experience on our mines has been that when a man goes on shift, the first thing he does is to see what the reading of this apparatus has been during the previous shift, so that he may keep up to a certain standard, or if possible, beat it.

The President: I am sure I can quite agree with Mr. Cullen's remarks. I have had cases come under my notice where a boiler plant has been fired with "that rotten Boksburg" coal, and has given excellent results. Then following that, "Middleburg" coal has been introduced, and absolutely no steam could be got; so that evidently the class of grate used in the boilers was unsuitable.

Prof. J. A. Wilkinson (*Member of Council*): I should like to ask Mr. McArthur Johnston if the CO_2 readings of the automatic recorders are checked from time to time by an actual analysis of the flue gases, which can be done in a very short time and with sufficient accuracy, by an Orsat's apparatus.

Mr. A. McA. Johnston: The CO_2 recorders are tested regularly by one of the members of our laboratory staff.

Prof. J. A. Wilkinson: It is also important that the amount of CO should be tested from time to time.

Mr. A. McA. Johnston: We test for CO_2, and as soon as we have the instrument in concordance with the Orsat, we run through several determinations of CO and then, if necessary, regulate the intake until no CO is found.

SUGGESTIONS FOR A NEW ATOMIC THEORY.

(*Read at April Meeting, 1909.*)

By James Moir, M.A., D.Sc., F.R.S.S.A.
(Vice-President).

DISCUSSION.

Mr. P. R. Roux (*Visitor*): To prove that a series of numbers occurs in regular sequence, put down half the numbers on a horizontal line in consecutive order, and the other half in a reverse order underneath, thus :—

3	6	9	12	15
30	27	24	21	18
33	33	33	33	33

It will be seen that the highest and lowest, second highest and second lowest, etc., give the same sum. Apply this law now to the atomic weights of the elements, keeping for the sake of simplicity to whole numbers. First note, that from hydrogen to tin, including nipponium there are fifty elements, which form the first half, consequently there will be many vacant spaces for numerical analogues in the second half, especially between bismuth and uranium.

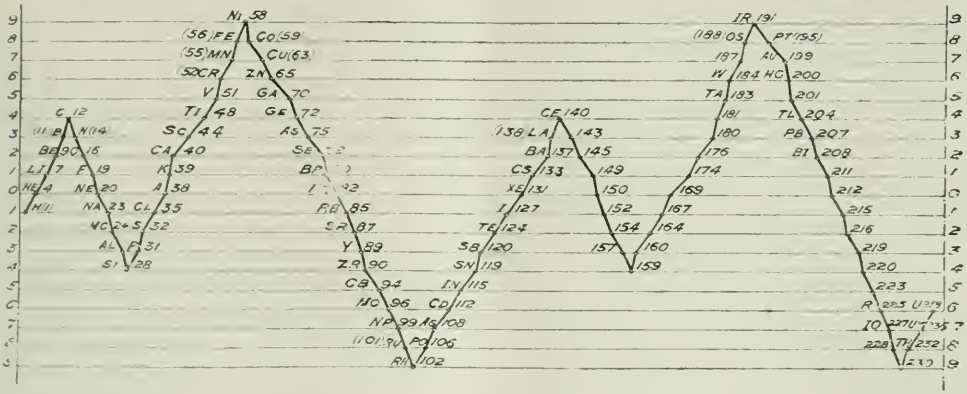
H 1	He 4	Li 7	Be 9
U 238	?Ux 235	Th 232	? 230
239	239	239	239
B 11	C 12	N 14	O 16
? 228	?Io 227	R 225	? 223
239	239	239	239
F 19	Ne 20	Na 23	Mg 24
? 220	? 219	? 216	? 215
239	239	239	239
Al 27	Si 28	P 31	S 32
? 212	? 211	Bi 208	Pb 207
239	239	239	239
Cl 35	A 38	K 39	Ca 40
Tl 204	? 201	Hg 200	Au 197
239	239	239	237

Sc 44	Ti 48	V 51	Cr 52	Mn 55
Pt 195	Ir 193	Os 191	? 187	W 184
239	241	242	239	239
Fe 56	Ni 58	Co 59	Cu 63	Zn 65
Ta 183	? 181	? 180	? 176	? 174
239	239	239	239	239

Ga 70	Ge 72	As 75	Se 79
? 169	? 167	? 164	? 160
239	239	239	239
Br 80	Kr 82	Rb 85	Sr 87
? 159	? 157	? 154	? 152
239	239	239	239
Y 89	Zr 90	Cb 94	Mo 96
? 150	? 149	? 145	? 143
239	239	239	239
Np 99	Ru 101	Rh 102	Pd 106
Ce 140	La 139	Ba 137	Cs 133
239	240	239	239
Ag 108	Cd 112	In 115	Sn 119
Xe 128	I 127	Te 127	Sb 120
236	239	242	239

Leaving out the more or less terra incognita between radium and bismuth, and again between lanthanum and cerium, we get twenty sets of analogues, comprising forty elements, to judge the law by. Of these forty 70% agree with the number 239, and the rest run from about 1·25% below to about 1·25% above the required number. I suggest that radio-active phenomena play an important part in the determinations of atomic weights. According to my law, gold should have a weight of 199 instead of 197. (Note, if one analogue has an even number the other has an uneven number.) During chemical action, when the atoms of gold are in a nascent condition, they would be far more liable to degradation than in a solid condition. And even in the solid state, when in contact with lead, particles of gold will migrate into the lead. Compare the analogous positions of Cu, Au, Ag and Ux on the diagram. The strong radio active properties of Ux are well known. It is also worthy of remark that these three metals are diamagnetic, the latter property suggesting the existence of emanations which repel magnetism. It would be interesting to put a definite weight of gold through chemical action over and over again, to see whether degradation does set in during a nascent condition. Oxygen also occupies an analogous position (third on the right of the apices of the triangles). And if the theory of degradation means anything at all, it certainly explains why oxygen is a fraction less than 16 if hydrogen is 1.

To digress for a minute, degradation is not quite a correct term to use. If the process, whereby the elements were primarily formed is termed evolution, then degradation is the right



word to use in radio-active phenomena, but the elements were never formed by a process of evolution, they were formed by an opposite process, that of involution. By involution, I mean a folding up—a condensation, and by evolution I mean an unfolding—an expansion. By looking at the question from this stand-point, it is clear that involution precedes evolution.* Involution is negative in character, and evolution is positive. There is one primordial substance permeating the universe in seven different forms—the seven fundamental notes that produce cosmic harmony—the solid, the liquid, the gaseous, the ultra-gaseous, the mind of man, the psyche and the primordial form. The process of involution proceeds to a given point in the circumference of a circle. then evolution sets in and continues to the starting point thus completing one of nature's endless cycles. Take uranium, it has reached the limit of involution, and since nothing remains in a stationary position for ever, it gradually evolutes up to the starting point—the primordial form—through the different stages of radio-active phenomena. During the first stage it breaks up into Ux and the difference in weight between the two soon migrates beyond the ken of man, where in course of time it becomes again subjected to the laws of involution.

We are now in a position to see that, however stable silver, gold and copper may seem to be in comparison to Ux, they are, nevertheless, subject to the same laws of radio-active phenomena.

Let us return to the atomic weights. The elements, when plotted in the manner shown on the diagram, produce a series of waves occuring in regular sequence, and vibrate across a common central zero line, upon which the inert gases are placed. This system reveals some most remarkable analogies of the elements at a glance. Hydrogen, the great stumbling block to periodists, fits perfectly into its place.

There remain three inert gases to be discovered. As the elements occur in perfect sequence one naturally expects a similar occurence of their properties, and this is exactly what happens. Their specific gravities reach their maximum at extreme points at carbon, silicon, nickel, rhodium, cerium, etc. As the elements approach the zero line from these points their specific gravities gradually decrease, except those of uranium and thorium. These two elements are only connected with the zero line in a reversed way, hence their exception to the general rule. Their melting points also follow each other in periodical sequence, but in a more complicated law. The maximum points are at or near carbon, silicon, titanium, iron, germanium, zirconium, rhodium, tellurium, cerium, etc., with a series of undulations between these points.

I have not gone into any further properties. The system is only a few days old, but I thought it would interest the audience to become acquainted with the ground I have covered so far. I fully expect that all other properties of the elements will show a perfect system depending on this one. Of course some of the atomic weights on the diagram are only approximate, consequently if chlorine be taken as 35·46 then its numerical analogue, thallium, must be 203·54, and so on with all the other pairs.

The President: I should like to thank Mr. Roux for his very interesting contribution to the discussion on Dr. Moir's paper.

ROUTINE ASSAYING ON A WESTRALIAN MINE.

(Read at December Meeting, 1908.)

By W. B. Blyth (Associate).

DISCUSSION.

Mr. H. A. White (*Member of Council*): I intervene in this discussion only for the purpose of knocking another nail in the coffin of that hoary headed superstition, the universal necessity for borax.

---

* I have explained this subject more fully in a small work that is still in manuscript form.

Mr. Wilmoth's contribution to this question reminds me of the celebrated painter whose effects were much admired by an enquiring student. Tell me, said the student, what do you use to mix your colours with? The reply was, "brains." Before I deal faithfully with this sinner I will read to you what Mr. Clennell, who is well and favourably known to many of us as a skilled experimenter, has to say upon the matter in the last issue received of the *Engineering and Mining Journal.* He says:—

"Experimental work carried on throughout a period of six months on a typical free-milling ore presenting no special features of difficulty, led me to form certain conclusions regarding the use of borax in assay fluxes. The investigation was undertaken mainly on the following grounds:—

(1) The fluxes previously used in assaying this particular ore had not given entirely satisfactory results.

(2) Statements had been made that the use of borax as an assay flux is liable to give rise to serious losses. It was thought worth while to test the latter point experimentally on a quartzose ore practically free from base metals except iron. The iron was present almost exclusively as oxide, in varying amount.

The chief points noted in the course of these tests were the following:—

(1) The use of a large excess of borax has a tendency to produce a hard stony slag very difficult to separate from the lead; when touched, especially with a sharp instrument, it sometimes flies to pieces with explosive violence or suddenly crystallises in radiating needles; a film of lead almost invariably breaks off the button and adheres to the slag. This detached lead can only be recovered with much labour and some portion is probably always lost in hammering out the buttons.

(2) When no borax at all is used in the flux, the button separates absolutely clean from the slag, so that in some cases it can be cupelled direct without even hammering out. While cooling in the moulds a scum rose to the surface, which on closer examination was seen to consist of a network of straw-coloured crystals, evidently resulting from some constituent of the ore which remained unfused in the absence of borax.

(3) The presence of borax made the charges distinctly more fusible and gave, usually, a clear greenish slag. By using a small quantity of borax in the charge, not exceeding 5 gm. to 10 gm. per A.T. of ore, it was found that a uniform fusible slag was produced without any noticeable amount of the scum, and that the buttons could be detached from the slag by a slight blow, without any portion of lead adhering to the slag.

(4) The slag became more deliquescent as the proportion of borax was diminished. This deliquescence is a disadvantage only when hammering out is delayed for some cause or other, as when a batch of assays is left in the moulds overnight.

In a series of tests, one assay of each sample was made with a flux containing no borax, and another assay with a flux as similar as possible, but containing borax. The fluxes were adjusted so that the total weight of the fusion charges and the quantities of charcoal were the same in each pair of assays.

Taking the results as a whole, it may be said that they show no marked advantage either way; they lend no support to the idea that borax carries any appreciable quantity of gold into the slag. Any losses of gold which may occur through the use of too large a proportion of borax seem to be due to mechanical causes, chiefly to the breaking off of small portions of the lead button in the process of hammering out, owing to the adhesion of the lead to the slag.

The addition of a small amount of borax seems, however, to be a slight advantage, perhaps chiefly on account of the greater fusibility which it imparts to the slag."

In this statement I may point out that the iron was present as oxide in his samples, and that he would probably have found slightly greater losses with sulphide ores, as I have pointed out before that borax does not favour the complete elimination of sulphur from the lead button. (Mr. Wilmoth may observe a possible source for some of his extreme cupel variations.)

The scum Mr. Clennell refers to is a misnomer, and he rather gives the impression that it is a part of the ore which has escaped fusion. I have here a sample, and you will note that it consists of bundles of needles—sometimes $\frac{1}{8}$ in. long, and they have obviously crystallised out from complete fusion.

I admit that borax hastens the fusion, and therein lies a danger, and you will always find its advocates keen on very fine grinding of their samples to avoid the case of apparently complete fusion when some quartz is left undissolved, as demonstrated by Mr. Atkin (vol. vi., p. 22). The mechanical losses experienced by Mr. Clennell, to which he refers, the small difference in favour of no borax, are illustrated by the specimen from a non-borax flux, showing how clean the lead comes off, and incidentally giving no signs of "an altogether useless scum-like mass," which is all Mr. Wilmoth is able to obtain with the same flux. I think I will leave him with the puzzle, "find the missing ingredient."

GRADING ASSAYS AND GRINDING EFFICIENCIES.

(*Read at December Meeting, 1908.*)

By Arthur Yates, A.I.M.M. (Corresponding Member of Council.)

DISCUSSION.

Mr. H. A. White (*Member of Council*): Mr. H. Stadler's contribution to this discussion breaks new ground and would entirely overthrow all previous calculations and work done in this subject unless some serious error has crept in somewhere in his conclusions.

I propose, therefore, to offer the following considerations for your examination, and I think the interests of this Society would be served if he were kind enough to give us a further contribution, exposing any fallacies I may have slipped into. Of course, the gist of the difference between the new and old methods lies in the "distance" factor which Mr. Stadler says has been cheerfully ignored. This is not exactly true, for the Rittinger method merely assumes this factor to be the same in all cases, and I hope to show the correctness of the assumption. Mr. Stadler assumes this "distance" to be a function of the diameter of the particle to be crushed, and in one method of crushing, at least, it may be granted that in effect this would be practically true. If, for example, compression be applied to opposite faces of a homogeneous cube of a material possessing elasticity of compression, the distance must slightly exceed that corresponding to the limit of elastic compression, which would evidently be "a fractional co-efficient of the diameter of the cube."

Also I cannot see how any method of actual crushing, even pure shear, can avoid this difficulty with homogeneous material, for shear can be shown to involve the same elastic factors. Of course, the fact that our ores are far from homogeneous renders the conclusion less certain, but we are driven to admit, that a certain amount of wasted compression or tension is involved in all practical cases. But if a given product is put before us, and we are asked to estimate what work is represented by it, I consider we are independent of how that work has been done, and our calculation must be based on the product, and not on its history, further than a statement of the size of particles commenced with. It is theoretically conceivable that a machine might be invented to apply power at the fracture plane itself and in that, the ultimate theoretical case, the work done must be estimated by the area of the fractures and the integrated force of cohesion multiplied into the distance through which the force of cohesion acts. In this case all the surfaces exposed represent fracture planes, and the increase of surface is exactly proportional to work done.

This is Rittinger's law, which is shown to be independent of any machine and is, therefore, the true theoretical criterion of work done. Any tension or compression elsewhere than at the fracture plane is only an accompaniment caused by the limitations of our machinery and must be ignored. If not, we may as well calculate the effective work done by a stamp by merely measuring its weight and drop, and it will be obviously absurd to do this if we reduced these factors below the point where any crushing at all is done, for in such case all energy applied is wasted in heat, and no efficient crushing is done at all on the rock.

THE PILGRIMS REST GOLD FIELDS AND MINING METHODS.

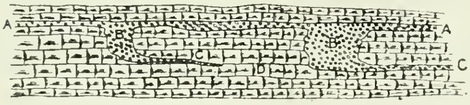
(*Read at March Meeting, 1909.*)

By J. Moyle-Phillips (Member).

DISCUSSION.

Mr. W. Price-Griffiths (*Member*): I read the author's paper on the above with a great deal of interest and pleasure. In a condensed manner he has dealt with very nearly all the salient points connected with mining in the Pilgrims Rest district, and his paper will no doubt prove of value to any who are, or will be, connected with these fields. There are, however, several points on which the author and I disagree, and these belong to the geological section of his paper. He states that the Theta reef is a coalescence of two or three different reefs. In my opinion, the Theta reef is one well defined interbedded sheet of ore, enriched and enlarged in places by one or two irregular deposits which cannot be called reefs. These irregular deposits occur along weaknesses in the bedding planes, produced probably by the folding of the strata. Thinning of the intervening strata brings these into closer proximity to the reef, with the result that metasomotic, or replacement agencies, have rendered the whole parting reefy. Thus we have an 18-in. reef merging into a body of ore 6 ft. to 10 ft. in thickness. This state of things may continue for several yards, but the same phenomenon is noticeable at the end as at the beginning. The central section of the ore body gradually changes into country rock, and the upper portion after increasing its distance from the reef proper, also fades in a like manner. This occurrence is quite common in most mines, but more especially at the Theta mine. At Clewer, what would seem at first to be two well-defined distinct deposits occur, but the top one only is persistent. The bottom one, it is true, exists in most of the explored area, but there are also many instances where it does not occur at all.

The accompanying figure illustrates an occurrence encountered quite recently in one or two drives in the Clewer mine, and seems to justify one in regarding the top reef as the parent deposit.



The author makes reference to the existence of a quartz leader, called the "Indicator," above the Theta reef. This is by no means general, in fact, I have seen it nowhere but at the Clewer. The term "indicator" is a misnomer and apt to be misleading. One would imagine from the name that it is as serviceable to the miner and prospector, as a finger-post on a cross road is to the traveller, whereas it is nothing of the kind. Everywhere the reef is the more prominent, and as a factor in determining the horizon of the reef, it is quite useless. Where rich ore has been exploited, the "indicator" has not been at all evident. "Indicators" are, doubtless, useful in fissure-vein mining, but with interbedded ore sheets they fulfill no useful purpose.

Another point on which I disagree with the author is contained in his explanatory notes to Fig. I. There he states that it is evident that the reef was formed before the dyke intrusions, as the latter penetrate the reef and appear on the surface. The relative age of the intrusions and the reef is a difficult question, and I fail to see where the reasons for the author's deductions are evident. From a perusal of the geological report of the Lydenburg district,* compiled by Mr. Hall, I find that that gentleman agrees almost word for word with the author on this same point. The penetration of the reef, which I take to be the author's chief evidence, does not, in my idea, support his contention. The reef occupies a certain geological horizon, and a certain position in the general arrangement of the strata, and from most theories that I have heard advanced on its origin, one is justified in assuming the existence in "pre-reef" days, of a certain strata or something which became the nucleus of the present mineral deposit. Now, does it not appear very possible that this nuclear layer was penetrated before the mineralisation had commenced? In the replacement theory advanced by Mr. Hall, the assumption of a nuclear layer seems to be absolutely essential, with which the circulating waters could "harder" their "goods," otherwise I fail to see why the reef should to-day occupy such a persistent position in the general sequence of strata. After the intrusions, the circulating waters would probably be more active, more mineralised, and their solvent power would be increased with the increase in temperature due to the intruding molten magma. On the Theta mine, large ore deposits have been found in the neighbourhood of huge masses of decomposed diorite of which the Nu paddock and the big opencast on the east side are in evidence. I should be very pleased if, in his reply to this discussion, the author will give some other reasons for his deduction, for as far as I can see, all the evidence favours an opposite theory.

From Fig. I. in his paper, the author seems to have an erroneous idea about the "strike" of strata. The section he gives is more along the "dip" than along the "strike." Evidently he is under the impression that the strike of the strata at Pilgrim's is east and west, whereas it is approximately north and south, the "dip" being west. The section fulfills the part it was intended for, *i.e.*, to show the undulating nature of the ground which constantly presents the management with innumerable problems in the economics of mining methods, and therefore, I consider it useless to dwell upon what may merely be a "terminological inexactitude."

Mr. Caldecott in his comments on the author's paper refers to the existence of "float" gold in the reef of this district. Although not extremely common, I have many times seen it myself when panning. It can generally be detected by its glitter on the surface of the water. I remember examining this gold once through a very powerful lens, and it presented a flaky sort of appearance, much resembling very minute thin shells of mica.

In conclusion, I should like to add my thanks to those already expressed by our President to the author for his very interesting paper.

THE INFLUENCE OF MOIST AIR ON QUICKLIME.

By Jas. Gray, F.I.C. (Member).

It is well known that quicklime on exposure to air is first converted into hydrate and then into carbonate, owing to the absorption of moisture and carbon dioxide.

---

* "The Geology of the Central Portion of the Lydenburg District," by A. L. Hall, Report of the Transvaal Geological Survey, 1906, pp. 73-100.

Veley (Transactions Chemical Society, 1893) deals with the absorption of carbon dioxide by quicklime, but I am not aware of the existence of any more recent literature on the subject. The matter was first brought prominently to my notice on comparing the percentages of available lime in the lime supplies during the winter and summer months, and I was surprised to find that the average percentage of available lime in the lime supplies during the rainy season were 15—20% lower than in winter, and especially so this year. It appeared improbable, under the circumstances, that this serious reduction in value was entirely due to the reversion of the lime into carbonate, and to elucidate the point the following experiments were carried out:—

10 grams of an ordinary commercial lime—containing 72·8% available lime—were placed in a platinum basin resting on a beaker of water, and a bell jar to prevent dust contamination placed over the apparatus, in such a way, however, as to allow free access of air. This exposure to an artificial moist atmosphere was continued for two days. After this period the increase in weight was found to be 11·35%, while the percentage of available lime had decreased to 63·78.

A similar experiment on the same lime, but continued for 10 days under similar conditions, showed an increase in weight amounting to 25·45%, and a reduction in the value of available lime to 48·44%. It was noticed in this experiment, as in others to be mentioned, that the increase in weight reached a maximum during the second day's exposure, and then gradually decreased, until after the sixth day, an increase of less than 1% was observed. The following is a typical example:—

		Grams.		%
Weight of lime taken		= 10·000		
,, after 1 day's exposure		= 10·5846	Increase	5·84
,, ,, 2 ,, ,,		= 11·3062	,,	7·21
,, ,, 3 ,, ,,		= 11·7998	,,	4·93
,, ,, 4 ,, ,,		= 12·0854	,,	2·85
,, ,, 5 ,, ,,	Av.	= 12·3760	,,	1·45
,, ,, 6 ,, ,,			,,	1·45
,, ,, 7 ,, ,,		= 12·4682	,,	0·92
,, ,, 8 ,, ,,		= 12·5450	,,	0·76

In view of these satisfactory results it was desirable to experiment on pure lime. A sample containing 91·9% CaO, the balance being $CaCO_3$, was exposed for six days under the same conditions as in the above experiments. At the end of that period the weight had increased 27·37% and the percentage of available lime decreased to 67·20, while 10·2% $CaCO_3$ was present.

A lime containing 11·93% $CaCO_3$ and 88·07% CaO, exposed for eight days in the artificial moist atmosphere, contained 16·38% $CaCO_3$ and 61·7% CaO.

It will thus be seen that the hydration of quicklime is extremely rapid, while the carbonation is correspondingly slow. In the two cases above quoted, after six days' exposure, the *increase* in $CaCO_3$ was only 2·1%, and for eight days 4·45%.

It may be argued that these experiments lose their value to a certain extent, owing to their having been carried out on finely powdered lime, a condition in which lime supplies are not despatched from the works, but the following experiment does not bear this out; indeed, the opposite argument would appear to carry weight owing to the larger surface exposed:—

Some large pieces of pure lime were strongly ignited until constant in weight, and they would then, to make a conservative estimate, contain about 95% CaO. After being exposed for eight days the percentage of available lime had decreased to 55·72, and the increase in weight was 34·11%, while the lime although retaining its form had become riddled with cracks, and was extremely friable.

From the above experiments the following conclusions can be drawn:—

1. That the absorption of moisture from the atmosphere by quicklime is extremely rapid, and, under the artificial conditions described, the lime is converted into $Ca(OH)_2$ in not more than six days.

2. That the absorption of carbon dioxide is very slow when compared with the rate of absorption of water.

It would appear that the usual guarantee of 75% available lime in lime supplies required by purchasers is, during the summer months, a hardship on the lime seller, as, notwithstanding satisfactory analyses of lime before despatching, by the time the consignment reaches the mine a large proportion of the lime has been converted into hydrate, causing an increase in weight and a reduction in value. The injustice done to the seller is obvious, and I am sure only needs to be pointed out to ensure a remedy.

Several methods to remedy the difficulty suggest themselves:—

1. All lime should be slacked before being despatched to the mines.

2. Seal the lime in air-tight cases (as with cyanide).

3. Weigh the lime on receipt at the mine, and carefully sample, to obtain a representative sample for analysis. Multiply the weight of the lime received by the percentage of available lime obtained by the analyst, and divide the result by the weight despatched from the works, which can be obtained from the railway note.

The last method appears to me the best, and the labour involved would not be so great, as a representative number of bags could be weighed, and their average weight applied to the whole consignment.

RESEARCHES UPON THE TELLURIDE GOLD ORES OF CRIPPLE CREEK (COLORADO).

By the Portland Metallurgical Society
(edited by Thos. B. Crowe).

Among the leading gold producing districts of the world the Cripple Creek district, of Colorado, stands as one of the foremost. This volcanic area, three miles long by five miles wide, embodies a network of gold-carrying veins, running in all directions, intersecting each other and comprising a series of ore shoots the magnitude of which is inconceivable. These ore bodies have in the past produced, and are now producing, a large tonnage of ore of exceptional richness. Along with the development and mining of these higher grades of ore material has simultaneously been developed, which, although of sufficient value to term ore did it exist in some other districts of the world, is on account of its refractory nature left standing in the mine stope or thrown upon the waste dump.

The refractory nature of these ores is due largely to the occurence of the gold in combination with tellurium, forming a compound $(AuAg)Te_2$, which is represented by the minerals calaverite, or sylvanite, and also to the association or encasement of these gold tellurides in crystal growths of the characteristic mineral pyrite.

The history of the metallurgical treatment of these ores is of great interest, and is conspicuous for the number of metallurgical failures which have taken place, due largely to the difference in the physical properties existing between gold telluride and native gold, and partly to the insolubility of the tellurides in cyanide solutions. Stamp milling followed by amalgamation was first tried; concentration failed, whilst at the smelting works these ores were not desirable owing to the high siliceous character. Only after roasting to free the gold of its volatile associate have the ores been susceptible to lixiviation processes. Consequently, roasting followed by barrel chlorination and concentration has become the popular treatment. This method works very well so far as the extraction is concerned, and will continue to be used so long as the present grade of ore is maintained; but as the amount of low grade ore standing in the stopes and upon the dumps becomes larger year after year, it becomes more apparent that a cheaper method of treatment must be found, the problem resolving itself into one of raw treatment at the mine, rather than transporting fifty miles by rail to the cheap fuel in the valley. With this end in view the Portland Gold Mining Company has carried on extensive experimentation, and we hope that a brief description of some of our experiences may be of interest.

Fine grinding in cyanide solutions followed by agitation and filter pressing was first tried, but difficulty in dealing with the stubborn sylvanite was experienced, so that our efforts were naturally directed toward the finding of a solvent for tellurium. We found, after experimenting along these lines, that the tellurium yielded to some extent under the action of oxidising agents, and after trying various acid mixtures we resolved to find an alkaline solvent; the most successful ones were the alkaline per-sulphates, alkaline hypo-iodites, and cyanogen iodide. The chemical behaviour of the alkaline per-sulphates is little known to metallurgists. They are not only solvents for tellurium, but their action when used in connection with cyanide solutions is extremely interesting. When thus used they act as slow oxidisers or depolarisers, thereby greatly increasing the dissolving power of cyanide solutions and when used in small proportions, 1 to 10 lb. per ton of solution, they do not destroy the cyanide to any great extent. We have done a good deal of theorising regarding the chemical action of these substances, especially when used in connection with cyanide solutions. When in solution alone, they act as strong oxidising agents, as follows:—

$$(NH_4)_2S_2O_8 = (NH_4)_2SO_4 + SO_2 + O_2.$$

But when mixed with cyanide solutions, their oxidising influence is greatly retarded. In view of this fact, together with results obtained in small tests, we are of the opinion that the per-sulphates, if produced at a low cost, could be generally used in cyaniding as oxidising agents. They are quite stable compounds and when mixed with working cyanide solutions would oxidise reducing agents, thus greatly aiding dissolving efficiency, precipitation, etc.

A recent article, "Cyaniding of Silver Ore in Mexico," written by Mr. W. A. Caldecott, of this Society, brings to mind a possible field for these substances as a means of dealing with the reducing agents, which this article states are a great source of trouble. We have also discovered that ammonium per-sulphate is a good solvent for silver, but to what extent this property can be applied in ore treatment has not been determined.

We have made the following few small tests on the solutions from our tailing mill in the endeavour to find out the effect of per-sulphates upon cyanide solutions when used as oxidising agents.

Three assay tons of blanket concentrates which assayed 108 oz. per ton were put in each of six bottles; in each bottle were poured 262 c.c. of our regular 1 lb. tailing mill solution; this made a three to one pulp, and 10 lb. of lime per ton of ore were added. To the first three bottles respectively, we added 0·1, 0·25 and 0·5 lb. of ammonium per-sulphate per ton of ore; the last three bottles being left as straight cyanide treatment. All the bottles were well shaken and stood over night.

	Cyanide consumed per ton of ore.	Ammonium per-sulphate added per ton of ore.	Solution Assay Value.
No. 1	·55 lb.	·10 lb.	4·13 oz.
,, 2	·55 ,,	·25 ,,	4·35 ,,
,, 3	·55 ,,	·50 ,,	4·46 ,,
,, 4	·60 ,,	None	3·54 ,,
,, 5	·55 ,,	,,	3·82 ,,
,, 6	·55 ,,	,,	3·72 ,,

From these experiments you can very readily see the effect of the increased dissolving efficiency obtained when an oxidising agent is used, the consumption of cyanide being practically the same as the straight cyanide treatment. Taking the average solution assay 4·313 oz. of the three bottles to which per-sulphate had been added, and comparing the average solution assay 3·693 oz. of the three bottles where no per-sulphate was added, and multiplying each average by three (as three to one pulps were used), we obtained a difference of 1·86 oz. extracted by the addition of small quantities of per-sulphate. You will doubtless be surprised at the low extraction obtained in either case, but it is a good example of the refractory nature of our gold. This material had passed through a roasting furnace, through chlorination barrels, over the Wilfley tables, then ground to 60-mesh in a 1 lb. cyanide solution, then caught on blankets, and was merely used in these experiments on account of its value to make a decided case.

As stated before, alkaline per sulphates are solvents for tellurium, and when mixed with cyanide solutions completely dissolve the telluride of gold. We have made several bottle tests on ore, using a solution of alkaline per-sulphate as a preliminary treatment, followed by a cyanide solution, the object being to first dissolve the tellurium, leaving the gold in a condition susceptible to cyanide solutions, but with poor results on account of reasons explained later, except where extreme amounts of per-sulphates were used. We have also tried a treatment using the alkaline per-sulphate and the cyanide together in the same solution, dissolution of the tellurium by the per-sulphate and the dissolution of the gold by the cyanide going on simultaneously, but with uneconomical results due to reasons explained later.

Alkaline persulphates, like some other oxidising agents, liberate iodine from potassium iodide in an alkaline solution, the liberated iodine combining with the alkali present to form alkaline hypo-iodites, which compounds we have found dissolved tellurium; we, therefore, conclude that the addition of potassium iodide to a solution of per-sulphate plus cyanide would form this tellurium dissolving compound, while the cyanide present would take care of the gold, the two working simultaneously.

For a long time we were led astray in our experiments. We had found that an excess of alkali in a solution of potassium cyanide plus potassium iodide plus alkaline per-sulphate greatly increased the solubility of gold leaf; test after test with good excesses of alkali was tried, believing that the solution that had the greatest dissolving efficiency upon gold, would have the greatest efficiency upon the telluride of gold, and working on the theory, that the liberated iodine formed hypo-iodite, which dissolved the tellurium, the cyanide present dissolving the gold, we could not see wherein an excess of alkali would be injurious; but we invariably failed to obtain a good tailing, unless we used an uneconomical percentage of per-sulphate. Finally, we discovered that cyanogen iodide was an exceedingly good solvent for the telluride of gold, and the idea occurred to us that it was possible to form cyanogen iodide in a solution of this kind by the action of the per-sulphate upon the potassium iodide, the liberated iodine combining with cyanide forming cyanogen iodide:

$$KCy + 2I = ICy + KI.$$

Knowing of the similarity existing between cyanogen iodide and cyanogen bromide, and having some knowledge of the cyanogen bromide process, we worked upon the theory, that the existence of cyanogen iodide depended upon the absence of excessive quantities of alkali. In following this out we found that we invariably obtained good tailings, even with very weak solutions. A solution of 1 lb. of cyanide, plus 1 lb. of potassium iodide, plus 3 lb. of alkaline per-sulphate per ton of solution, in a pulp of three solution to one of ore, with 4 lb. or 5 lb. of lime per ton of ore, gave us on 1 oz. ore, ground to 100 mesh, tails of less than $1, with a consumption of cyanide of about 1 lb. per ton of ore. But we found that our solutions after leaving the ore and standing in contact with the air, lost their cyanide, a very hard thing to believe, as the same solution before being applied to the ore would stand in a beaker indefinitely with a very little loss of cyanide. Even when in contact with the ore in a bottle, this loss was not excessive, but when separated and left to stand, the cyanide in some cases disappeared rapidly. We discovered that this loss was more pronounced when treating heavy sulphide ores; and we are led to believe that it is due to the formation in our solutions of $Na_2S_4O_6$. Sodium thio-sulphate, $Na_2S_2O_3$, being formed by the action of the per-sulphate upon the pyrite, is probably further oxidised to $Na_2S_4O_6$.

$$2Na_2S_2O_3 + I = 2NaI + Na_2S_4O_6.$$

When the solutions are in contact with the ore, the reducing action of the ore aids in overcoming oxidising influences, but when separated, the oxidation becomes more violent.

The action of the air seems to have a marked influence on the cyanide destruction, as at any rate, a decided difference in consumption is noted in using closed and open agitators. However, we find that on passing the solution through the zinc boxes, the deterioration of the cyanide is stopped, due undoubtedly to the reducing action encountered there.

We have experimented in a small way upon the manufacture of sodium per-sulphate by the electrolysis of salt cake, and obtained a product which, when used in bottle tests, answered very well, so that it might be produced on a large scale at a reasonable figure.

Potassium iodide, although an expensive chemical, would remain as such in the solution, the per-sulphate causing it to give up its iodine, this iodine combining with the cyanide to form cyanogen iodide, or with the alkali to form an alkaline iodide, or alkaline iodate, but always finally reverting back to an alkaline iodide, as any iodate formed would be reduced to iodide in the zinc box. Cyanogen iodide is certainly a wonderful solvent for gold tellurides, and its production by this method we believe presents a possible field for ore treatment. We understand that in the practice of using cyanogen bromide as a solvent, one of the troubles other than its cost, is its rapidity of action, as it remains as cyanogen bromide only for a limited period, and does not act sufficiently long to cause the dissolution of the telluride of gold. You can, therefore, readily see that if it were possible to produce it in a similar manner to the one described, a much weaker solution constantly in contact with the ore, would have a decided advantage.

In an attempt to present this to you in a more lucid manner we add the following summary; our object being to treat the finely crushed ore by one of the following schemes:—

1. A preliminary treatment by an alkaline persulphate, followed by a treatment with a cyanide solution.
2. A treatment by a solution containing both alkaline per sulphate and potassium cyanide.
3. A treatment by a solution of an alkaline persulphate, an alkaline iodide, and potassium cyanide, contained in one solution.

In the first, we find that it requires a strong solution of persulphate to obtain a good tailing. In the second, our object was to produce an alkaline hypo-iodite, by so doing we would have an alkaline persulphate and an alkaline hypo-iodite acting as tellurium dissolving substances, and potassium cyanide acting as a gold dissolving substance, but we have found that it requires a strong mixture of the above to obtain the extraction. In the third, our object was to produce cyanogen iodide, and not to depend directly upon any of the ingredients other than cyanide and cyanogen iodide, as solvents.

As stated in the first portion of this paper, the refractory nature of these ores is due largely to the occurrence of gold in combination with tellurium, etc., and also to the association or encasement of these gold tellurides in pyrite. We had found that with any of the three schemes our dissolution was perfect, even with weak solutions when working upon crystals of telluride of gold, but when working on ores where part of the values was associated with the pyrite, we experienced difficulty with the first two schemes; but the third invariably gave good tailings. Concentrates are usually a very refractory material to handle, even when the gold occurs as metallic gold associated with the pyrite, but as in this case we have instead of metallic gold, the refractory telluride of gold associated with the pyrite, you can readily appreciate the problem which confronts us; although the oxidising agents, persulphates and hypo-iodites as used in the first two schemes act very well upon the crystals of sylvanite; when coming in contact with the sylvanite only; associated with the reducing pyrite, their function is to a great extent destroyed.

In presenting to you these experiences we hope that we are able to lay before you food for thought, which may ripen into fruit of benefit to metallurgy; but we realise that this fruit is far from mature, and therefore, solicit your co-operation and views upon the subject.

The meeting then closed.

Contributions and Correspondence.

THE PRECIPITATION OF GOLD FROM SOLUTION BY CARBONACEOUS MATTER.

A description of a reliable method for estimating the unburnt carbon in lime may be of use to some of our members. Ten grammes of the crushed lime are slaked with water and then treated with dilute hydrochloric acid in a porcelain basin. The solution is boiled for a few minutes to dissolve the lime and drive off all CO_2. The liquid is then decanted from the undissolved portion. This undissolved portion, containing the carbon required, is then washed by decantation with water till the washings are no longer acid, and the residue is dried at a gentle heat. This residue is mixed with a little finely ground cupric oxide, freshly ignited, and the mixture transferred to a small porcelain boat. The boat is then placed in a piece of combustion tubing about 10 in. long

A tin box, 5 or 6 in. long, about 4 in. wide, and 4 in. deep is lined with asbestos millboard. In the centre of each end a circular hole is cut sufficiently large to allow the combustion tube to pass through, and another hole about 1½ in. in diameter is cut in the bottom. The top of the box—conveniently a large oblong cigarette tin—is covered with a piece of asbestos millboard when the apparatus is in use. The tube containing the boat is fitted into the tin and a current of air is drawn through the tube by means of an aspirator. This air is first passed through a wash bottle containing caustic potash, then through a drying tower containing calcium chloride. After that it is passed through the combustion tube, the boat being heated to redness by means of a bunsen burner or petrol flare lamp, the flame of which comes through the hole cut in the bottom of the tin.

The carbon burns off and the products of combustion pass, first through a U tube containing calcium chloride, and then through weighed potash bulbs. Between the potash bulbs and the aspirator a guard tube containing, soda lime and calcium chloride, is inserted. After 20 minutes' heating the tube is gradually cooled down, and the air current stopped, after aspirating through enough air to drive all the CO_2 out of the apparatus. The potash bulbs are then detached and weighed, the increase in weight multiplied by 0·2727, giving the amount of carbon present in the 10 gm. of lime.

I have found the asbestos lined tin box a very good substitute for a small combustion furnace, and when once the apparatus has been fitted up, the estimation can be done in about an hour. The results are more reliable than those obtained by an indirect method. A fused silica tube will last much longer than one made of hard glass, and will perhaps be cheaper in the long run.

F. W. Watson.

Germiston, April 25, 1909.

ASSAY OF SLIMES RESIDUES.

A little time ago Mr. Rusden read a paper before this Society on the discrepancies obtained in assaying slimes residues. The subject is of considerable interest as we have to rely upon the results obtained by assay, almost entirely, not only in forming a judgment in regard to rival processes, but also in carrying out our daily work. The introduction of the Adair Usher process on some of our mines has brought in another factor to be considered, in the shape of difficulty in obtaining accurate or representative samples of the slime thus treated. This process is certainly elutriative in action and the heavier portion of the charge, consisting of fine sands and pyrites selectively settles on the bottom of the tank, with a gradation to the finest and lightest at the top of the settled charge. With cone bottomed tanks this separation is more accentuated, and as it is now customary to erect these in very large sizes, say up to 56 ft. in diameter, and without any means to reach the centre, the rod sampling, formerly in use, becomes unreliable. This brings us back to the alternative of sampling the residue at the discharge pipe, which gives a sample with a large proportion of liquid; and thus the trouble shown by Mr. Rusden in assaying residues containing gold in solution, is accentuated to a greater degree. The trouble is a real one and though it can be overcome to a large extent by slow evaporation in porcelain dishes, this method is cumbersome and troublesome in the care it needs. I therefore make the suggestion of combining some of the ideas previously given by members of the Society as follows :—

1. Obtain the sample by a slotted pipe in the discharge pipe.
2. Treat the sample by the copper method of precipitating the gold, which will give all the gold in the solid portion.
3. Separate the barren liquid in the air pressure sample filter, which is in general use for washing slimes residues.

The result is a sample containing at most 35% moisture, which can be dried without any special precautions, except that it is advisable to pass through a fine sieve after drying, in order to ensure that is well mixed.

I am not yet prepared to present any results obtained, as I should prefer to perfect some of the details before so doing, but I take this early opportunity of making the suggestion in order that those interested may be able to try it for themselves in the meantime.

S. H. Pearce.

Johannesburg, May 7, 1909.

With reference to the above communication, I am pleased to note that tardy recognition has been given to the method used by me as far back as June, 1902, for the assay of slimes pulp residues.

One month's experience of the simple evaporation method was sufficient to convince me of its inadequacy, and it was only on the advice of my principal that I then refrained from publishing my results, and giving the same method now recommended by Mr. Pearce. If I remember correctly, Mr. Pearce was made acquainted at the time with the success of my process, which, of course, is merely a natural sequent to my published method for the assay of cyanide solutions,* and spoke highly of its possibilities.

It will be within memory that Mr. Caldecott so far back as July, 1898, met with undeserved criticism in the discussion on his paper, "Discrepancies in Slimes Treatment," and Mr. Torrente again had occasion to refer to these discrepancies in August, 1904, whilst, if I am correctly informed, the authorities of the Eckstein Group of Mines circularised all their assayers on the subject shortly after Mr. Rusden's research. There has, therefore, been no lack of data to go on, and it is very much a matter of surprise to me that Mr. Pearce has made no reference to my personal connection with a controversy which has raged for so long. Apart from private dissemination of facts I must refer him to my paper on "Routine Assaying," vol. vi., p. 27, and the discussion on Mr. Torrente's paper, "Improvements in Slimes Treatment," vol. v., pp. 127-8.

It would hardly be too much to say that some of the gold supposed to be annually stolen might very well be accounted for by the discrepancies between theoretical and practical extraction from slimes.

I can show figures giving losses as high as 70% of the total gold in slimes pulp residues by simple evaporation, and in practice using the greatest care the loss seldom fell below 30%. In all cases the greater loss was concomitant with a high solution as against a low insoluble gold value, indirectly a justification for treating the pulp as a simple solution.

A. Whitby.

Johannesburg,
May 31st, 1909.

Visit to Simmer Deep, Ltd.

Nearly 200 members of the Society visited the Surface Works of the Simmer Deep, Ltd., on Saturday afternoon, the 22nd May. On their arrival on the property they were received by Mr. F. C. Frey, the Manager, Mr. H. C. Behr, the Consulting Mechanical Engineer, Mr. W. A. Caldecott, the Consulting Metallurgist, Mr. A. McA. Johnston, the Acting Reduction Works Officer and other members of the staff, and shown over the plant, of which the following is a description kindly furnished by Mr. Jas. E. Thomas, the Reduction Works Manager :—

These Reduction Works were erected and put into operation on 1st September, 1908, to treat the ore from these two companies, both of which are under the control of the Consolidated Goldfields of South Africa, Ltd.

Separate mill bins of 5,460 tons and 2,525 tons capacity are provided, the ore from each mine being treated separately until it leaves the mill tables, when it mixes on entering the tube mills and thence the cyanide plant. The recovery obtained by tube milling and cyaniding is apportioned to each company on the basis of the tonnage and value of the tailings leaving each company's mill tables.

Briefly, the processes are as follows :—

Breaking the ore to $1\frac{3}{4}$ in. cube at each of the company's sorting and crushing stations ; transport by means of hopper-bottomed cars of 35 tons and 45 tons capacity, drawn by a 48 ton locomotive on a 42 in. gauge track ; stamp milling with heavy gravitation stamps, of the Californian type, and amalgamation over stationary copper plates ; tube milling and amalgamation over copper shaking tables ; classification of slimes and sands by means of cone classifiers ; treatment of sands by means of rotary filter tables, and wet filling of tanks with cyanide solution, with subsequent transfer and percolation with cyanide solution ; treatment of slimes by the decantation process ; precipitation of gold from solutions by zinc shavings, zinc-lead couple ; acid treatment of gold slimes, calcining and smelting, etc.

The mill consists of 300 head, arranged five in each mortar box, in blocks of 10, each 10 stamps being driven by a 50 h.p. three-phase electric motor. The current is generated and supplied by the Victoria Falls (Transvaal) Power Co. from their station at Brakpan, about 20 miles distant.

The framing of the mill buildings and bins is entirely of steel girders, the bins themselves being of timber 6 in. thick on the sides and 6 in. at the ends. The mortar boxes are of the straight-backed type, and are specially heavy, having a bottom 15 in. thick. Each box weighs 11,872 lb. These are placed on concrete foundations—separate foundations for each 10 stamps, 17 ft. × 10 ft. at the base and 15 ft. × $4\frac{1}{2}$ ft. at the top and 10 ft. high. Each block contains about 102 tons of concrete, and has a sheet of best rubber $\frac{1}{4}$ in. thick between the bottom of the mortar box and the concrete. The holding down bolts are six in number, 1¾ in. diameter by 7¾ ft. long, so placed as to be readily accessible for tightening and replacing. The mortar boxes are 6 ft. high, taking a 59 in. screen frame. The height of the screen opening is 24 in. Each box has five 1 in. openings for the admission of water from the back water service. These are arranged so as to give a jet of water playing on each die at an angle of 45° to the surface of the die. Manganese steel liners are fitted in each box to take the wear due to attrition. The stamps weigh 1,670 lb. each when new, made up as follows:—Stem, 723 lb.; tappet, with gibs and keys, 252 lb.; head, 410 lb.; shoe, 285 lb. The stems are 4 in. diameter, and are placed at 10¾ in. centres. The order of the drop is 1, 3, 5, 2, 4. The set height 8½ in. Height of discharge, 4 in., and number of drops 100 per minute. The estimated duty per stamp is 8·7 tons per 24 hours, but this has already been exceeded, using a 500 mesh screen, 0·033 in. aperture. The king posts are of timber of the "built-up" type, to prevent twisting, and sit in cast iron shoes resting on the rubber on the concrete foundations. This also obviates having an unnecessary amount of timber above the cam shaft bearings. The cam shafts are 16 ft. 3 in. long and 7 in. diameter and the majority are of the "Riffled Blanton" type. Other methods of fastening the cams on the shafts are also being used.

---

* Vide vol. iii., pp. 6 and 9.

The feeder chutes are provided with sliding doors on the bins, whilst the feeders themselves are driven by means of ¾ in. Manilla ropes from rocker bars placed at the back of the king posts above the cam platforms. The front of the cam platform is supported directly from the concrete foundations, independently of the King posts. The reduction of the vibration on the platform, due to this method, is very noticeable.

The mill clean-up room is very spacious, and contains three revolving drums for cleaning amalgam, three amalgam barrels, three bateas, and a clean-up table, sump, etc. Besides these, a small tube mill, 6 ft. 6 in. × 5 ft., with a shaking table; two 10 ft. × 10 ft. conical bottom vats, and a zinc precipitation box, for the treatment of black sands, barrel tailings, etc., by tube milling and cyaniding, are provided. Two retorts and three Cornish fires are also in this room, so that the amalgam is retorted and the bullion run into bars before leaving the mill buildings.

The carpenter's and mill fitter's shops are under the same roof as the clean-up room, but are separated therefrom by a fire-proof brick wall. The only entrance to the clean-up room is from the mill, past the mill foreman's office. The arrangement of the batteries back to back, 150 stamps each side, with the bins between, admits of extensions being easily made at the southern end of the mill. The motors drive down to small counter-shafts, one for each 10 stamps (which are placed below the feeder and motor floor), by means of 11 in. belts. Thence the drive goes direct by means of a 21 in. belt to the 7 ft. cam shaft pulleys. The counter-shafts are moved by means of bevel geared wheels, so as to take up the stretch in the motor and cam shaft pulley belts.

On leaving the mill tables the tailings pass through mercury traps and launders to the tailings pits, of which there are two, one for the 200 Simmer Deep, and one for the 100 Jupiter stamps. From these the pulp is elevated by means of 8 in. centrifugal pumps, three to one pit and two to the other (only one in each pit being normally run at a time), to the cone shaped tube mill classifiers. Of these there are two to each of the four tube mills, 45 in. in diameter at the overflow, and 7 ft. deep, with a ⅞ in. nozzle at the underflow. These in turn deliver into a de-watering cone, 36 in. diameter and 5 ft. deep for each tube mill, with a 1¼ in. nozzle at the underflow. The overflow from these de-watering cones joins the stream at the tube mill outlets so as to make the reground pulp fluid enough to pass over the shaking tables, of which there are five, 12 ft. × 4 ft. 7 in. to each tube mill, and which run at 200 shakes per minute. The tube mills are of the "Krupp" type, and are 22 ft. × 5 ft. 6 in. inside the shell, and are lined with 5½ in. local flint setts. Each mill is driven by a separate 125 h.p. electric three-phase motor and running at 30 revolutions per minute, takes about 104 h.p. The pulp passes from the underflow of the de-watering cones through "Pryce's" feeders, through which are also fed the pebbles to maintain the pebble load in each mill about 6 in. above the centre of the mill. These pebbles are pieces of ore about 4 in. cube, picked off the belts of the sorting stations and delivered to a special bin from which they are trammed to the tube mills in small cars of 10 cubic ft. capacity. The average working load of pebbles in each mill is about 10 tons, and about 5 tons per mill per day are fed in to maintain this.

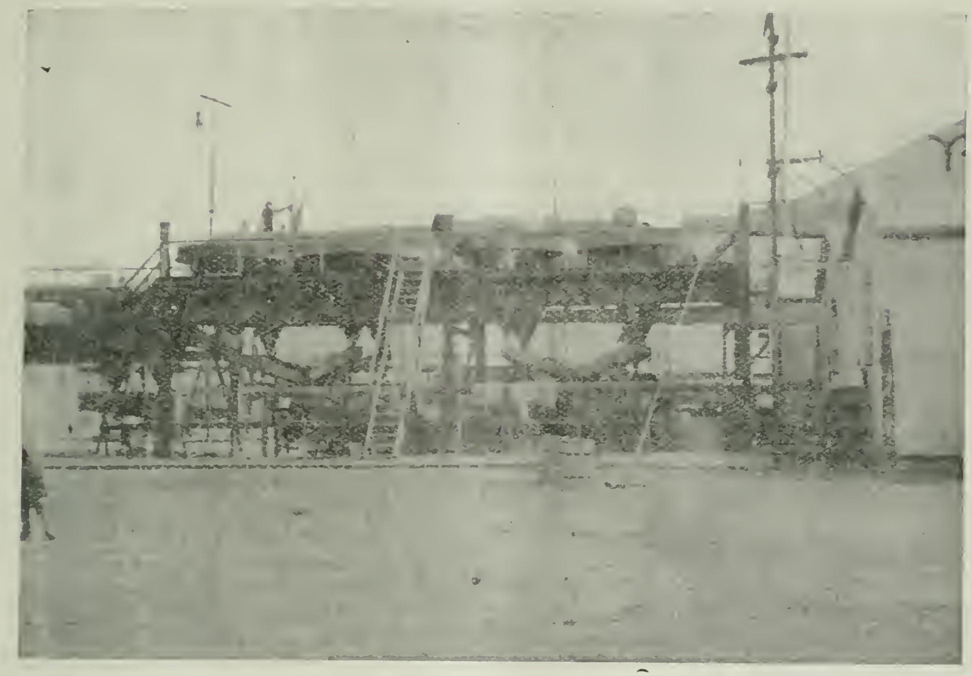
The tailings from the shaking tables join the main pulp in the tailings pits and are re-elevated to the classifiers, any particles that still require regrinding passing down again through the classifiers, the remainder overflowing with the fine product in the mill tailings, to the cyanide works. A small locomotive type boiler is installed in a separate shed, outside the mill buildings, for providing steam for steaming plates, etc.

Hand samples of the pulp entering the cyanide works are taken every hour at the overflow launders of the tube mill classifiers and reserved for grading and assay. The pulp is then run to

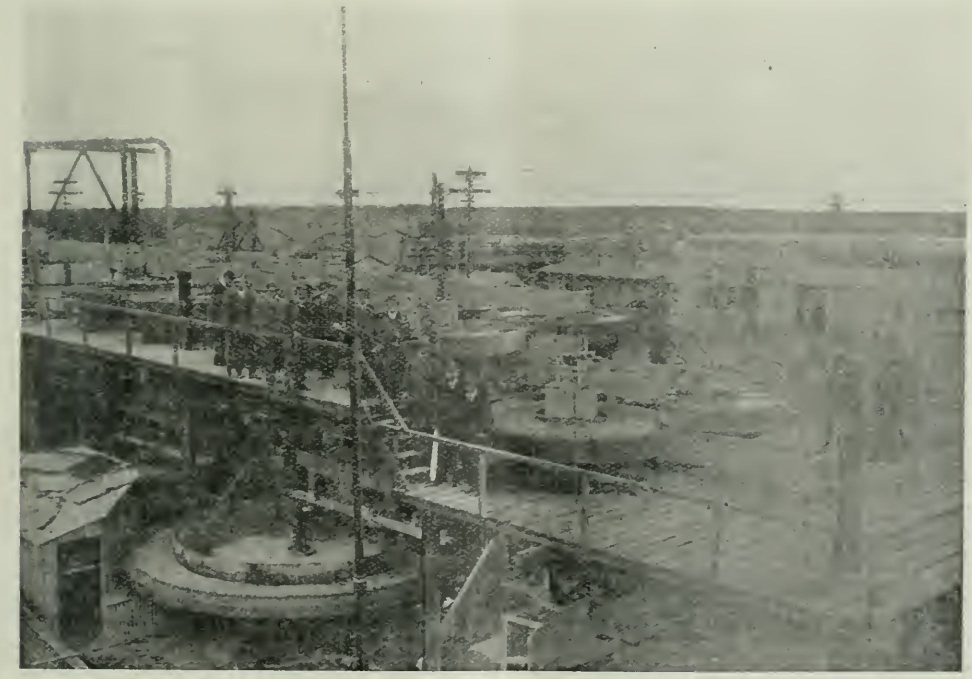
12 cone classifiers, placed in parallel, 6 ft. diameter at overflow and 6 ft. 6 in. deep with a 2½ in. nozzle and internal regulating plug at the underflow. The pulp overflowing these classifiers runs directly to the slimes plant, passing over ten return sands spitzkasten en route. The underflow, containing about 4% slimes, is pumped by means of a 6 in. centrifugal pump to the cones of the continuous sands filter plant, with the addition of a little clear water. This plant consists of two rotary filter tables, 20 ft. diameter, with a 2 ft. 6 in. filter bed, equal to 137 sq. ft. each, revolving at a speed of one revolution every three minutes. The filter is composed of strong screening, supported on wooden slats on edge, 4½ in centres, over which is laid coir-matting well caulked at the inner and outer edges. On this again are placed strips of unbleached calico to prevent any slimes which may penetrate through the working bed of sand, 1½ in. thick, choking the coir-nut matting. Over each of these tables four cones, 8 ft. diameter at the overflow and 8 ft. deep, are placed. The pulp overflowing, after passing over two return sands cones of the same dimensions, joins the main stream to the slimes plant. The underflow, containing about 30% moisture and 1% slimes, runs down launders of 30% grade on to the filter beds. The underside of this bed is connected to a vacuum pump, through a receiver, the moisture collecting in which is removed by means of a three-throw plunger pump and returned to the main tailings stream. A vacuum of from 5 in. to 10 in. is found most effective, if the vacuum gauge reads higher, it shows that the sand of the filter has become choked with slimes and requires renewing. This is usually done once every 24 hours, the operation taking about 50 minutes for both tables. Only three cones are normally in use for each table, the fourth being only used when the other table is out of action for renewal of filter bed, etc. One table, working with four cones, is capable of dealing with about 60 tons of sands per hour for four or five hours. Both the primary and secondary cones are provided with diaphragms placed about 2 ft. above the underflow. The sands, containing about 15% moisture, are removed from the table by means of a stationary plough and falls into a launder in which a 0·03% KCy solution is running. Thence it is elevated by means of a centrifugal pump to the collecting vats. Of these there are eight in use, two being utilised as storages for solution, and the other six as sand collectors proper. These vats are 50 ft. diameter, and 8 ft. 3 in. in height above the filter mat, and are fitted with "Butters and Mein" distributors and annular overflow launders. The solution overflowing the vat that is being filled runs into one or other of the storage vats from which it gravitates back to the launder at the rotary filter tables, and is again pumped up with fresh sands from the tables. Means are provided for moving any accumulation of slimes in the storages to the slimes plant for treatment. Each collecting vat averages 738 tons of dry sands. After being filled and leached dry, the charge is removed through ten bottom discharge doors on to three shuttle belts, which in turn discharge on to the main belt. These shuttle belts are mounted on rails and are so arranged that they can be readily shifted from one vat to another by means of a traversing gear. Altogether there are six shuttle belts, 42 ft. centres and 32 in. wide for discharging the collecting vats so that while three are being used, the other three can be placed in position under the next vat to be discharged, and so save time. The time required to empty a collecting vat of sand is three and a half hours, the shovelling, etc., being done by twenty-four natives. From the main belts—of which there are two—one 275 ft. centres and 28 in. wide, and the other 453 ft. centres and 28 in. wide, driven by two separate 20 h.p. and 50 h.p. motors, the sand passes to another shuttle belt, 260 ft. centres and 32 in. wide, running on staging between the treatment vats, 50 ft. × 10 ft., arranged in two rows of five. From the shuttle belt it is then discharged to "Blaisdell" distributors, two in number, each running on rails over each row of treatment tanks. The sand is discharged from these after completion of treatment by means of 20 cub. ft. trucks fitted with side jockeys and wire rope mechanical haulage to the two residue dumps.

The slimes plant is of the decantation type, and consists of four collecting vats, 70 ft. × 12 ft. deep at the sides with a cone 5 ft. 6 in. deep, equal to a flat bottom vat 70 ft. × 13 ft. 10 in. deep. There are also four first-wash vats, one intermediate transfer vat, and four second-wash vats of the same dimensions. Each vat is capable of holding 400 tons of dry slimes and 1,400 tons of solution, and is fitted with a centre valve which also contains a 2½ in. nozzle for solution to facilitate transferring and discharging of the slimes pulp. This is accomplished by means of two 12 in. centrifugal pumps, with 16 in. suction and discharge pipes. Only one of the pumps is used on any one vat at a time, but they are so arranged that either pump can be used on any vat for one operation, while the other is being used for another operation on any other vat. The first-wash solutions are decanted by gravitation to a small vat, 20 ft. × 6 ft., fitted with an annular overflow launder. This allows of the settlement of calcium carbonate from the solution, which would otherwise choke up the cloths of the

CALDECOTT'S VACUUM FILTER TABLES AND CONES.



Side View.



Top View.

clarifying presses. The solution is pumped from this vat by a 6 in. centrifugal pump through three clarifying presses to a steady-head tank, thence to the extractor boxes. The second-wash solution can also be run through this system to the extractor boxes, if desired, but is normally pumped through a 6 in. centrifugal pump to the first treatment vats or intermediate second-wash storage as required. This is of the same dimensions as the precipitated slimes solution intermediate storage, viz., 70 ft. × 12 ft., and is placed alongside it. The solution from these storages is pumped through either of two 8 in. centrifugal pumps with 12 in. suction and discharge pipes, to any of the nine treatment vats. A separate 6 in. centrifugal pump is provided to deal with the water decanted from the slimes collecting vats. This water joins the stream from the annular overflow launders of these vats and runs to the return water vats, one 70 ft. × 16 in., fitted with an annular overflow launder delivering into a second, also 70 ft. in diameter and 12 ft. deep. This minimises the risk of slimes remaining in suspension in the water returned to the mill. From the return water tanks the water is pumped by three 8 in. centrifugal pumps to the mill service tanks. Of these there are two, one 40 ft. × 14 ft. and the other 40 ft. × 12 ft., mounted on staging 22 ft. high, and arranged in the same manner as the return water vats. Outside the extractor house is a shed containing one vacuum and one three-throw plunger pump used to assist leaching and to aerate sand charges whilst under treatment in either the collecting or treatment vats.

The extractor house contains twenty steel zinc boxes, 30 ft. long and 5 ft. wide, each divided into two boxes of six compartments. Twelve boxes are placed on the north side for dealing with solutions from the treatment of the sands. Of these, two are used for strong solution, over 05% KCy, the remainder being for weak solutions. One of the latter can be used for either weak or strong solutions, as desired. The other eight boxes are placed along the west side of the extractor house, near the clarifying presses, and deal with solutions from the slimes plant only. Each of the clarifying presses contains 48 hollow frames, 32 ft. × 32 ft., and gives 600 sq. ft. filtering area, and is placed in a shed just outside the west side of the extractor house.

The clean-up room for acid treatment of the gold-bearing zinc shavings from the extractor boxes, is in the middle of the extractor house, the zinc cutting lathes occupying the east side, and the refinery the south side of the building. The extractor boxes, clean-up room and refinery are fenced in, so that no unauthorised person may enter. The acid treatment plant consists of three dissolving vats, 10 ft. × 6 ft., and three washing vats 10 ft. × 10 ft. These are all provided with mechanical stirring gear. Bi-sulphate of soda is used for dissolving the zinc from the boxes. This is dissolved in water in a lead-lined vat, 10 ft. × 10 ft., so as to give a solution containing about 18% free sulphuric acid. This is diluted with its own bulk of water before being used in the zinc dissolving vats. The bisulphate dissolving vat is placed at the south end of the shed containing the clarifying presses, with a small boiler of the locomotive type between. This latter is used for heating the bisulphate solution when necessary and for obtaining steam and hot water for the various small jobs, such as washing the cloths of the clarifying presses, washing the acid treated gold slimes in the filter presses, etc., etc. These filter presses, of which there are three, 24 in. × 24 in., stand in the same room as the acid treatment vats. The washes decanted from the acid washing vats, flow into two wooden vats, 25 ft. × 12 ft., where they are allowed to stand for several days and, if necessary, treated with zinc dust before being run to waste. The solution from the strong extractor boxes runs into two steel storages, 40 ft. × 12 ft., that from the weak boxes into two, 50 ft. × 12 ft., and the solution from the slimes boxes into one storage, 50 ft. × 12 ft. Another storage of the same dimensions is also provided for surplus solution, *i.e.*, last drainings from the sands treatment vats, low in gold values, waiting to be transferred to the slimes plant for the sake of its free KCy.

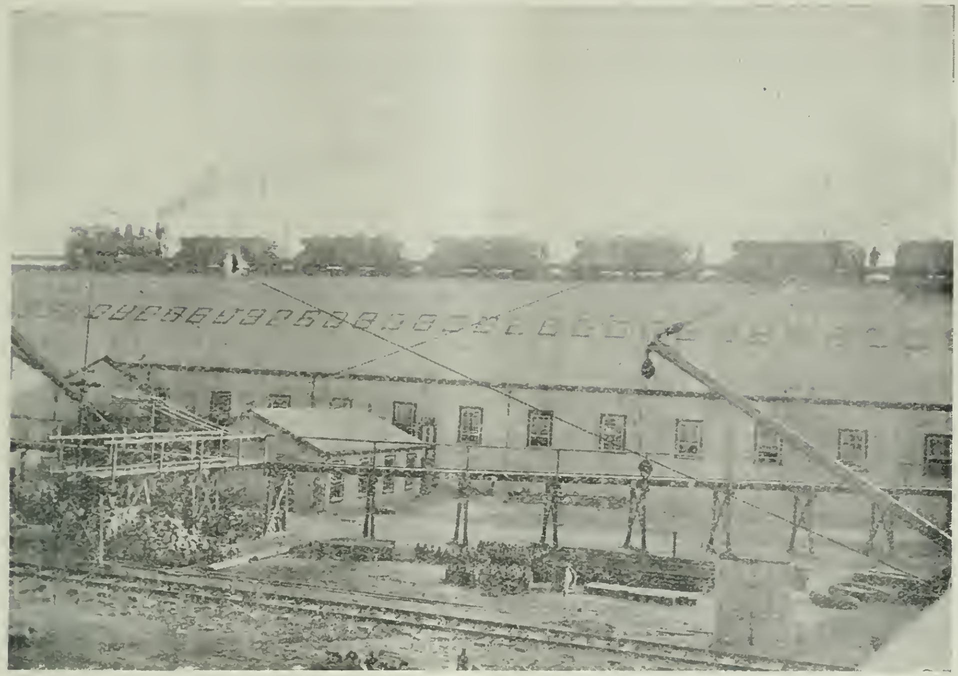
The refinery, which occupies the south side of the extractor house building, contains, besides a fluxing room, one reverberatory pot furnace, capable of holding twenty six No. 100 pots; one calcining furnace; one pan furnace for smelting slags, sweepings, etc., etc. One cupel furnace and three Cornish fires for melting and casting the bullion into bars.

As at the mill, no gold bearing material leaves the extractor house building except in the form of bars of bullion ready for depositing in the bank.

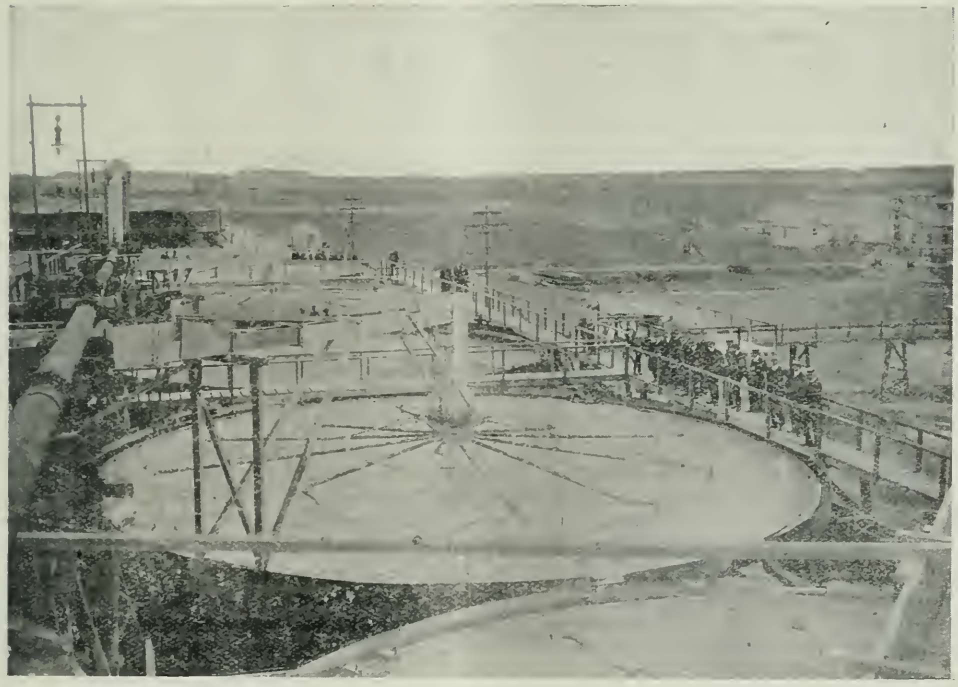
All the furnaces in the refinery lead into one main flue, 3 ft. × 2 ft. 6 in., ending in an iron stack 90 ft. high by 5 ft. 6 in., lined for 40 ft. of its height with fire brick.

The party were afterwards hospitably entertained by the management. The President (Mr. R. G. Bevington) expressed the thanks of the members of the Society present for the opportunity given them to visit and inspect one of the most up-to-date gold reduction plants in the world.

Mr. H. C. Behr suitably responded.



Simmer Deep and Jupiter Joint Mill.



Sands Collecting Tanks.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

BOILER FEEDWATER AND ITS TREATMENT.—"There are two general courses open in the matter of the treatment of feedwater for boilers: (*a*) precipitation in the boiler; (*b*) the purification of water by softening. In the past, and even at present, the former method is being pursued in the mining sections of the Pittsburg district, while the latter method has been adopted by the leading manufacturing industries in the immediate vicinity of Pittsburg. Of course, the water-softening method is the logical way in which to cope with bad water, and is the only course open if entirely satisfactory results are to be obtained. However, since it has been the practice, and is at present, for coal and coke operators to treat the feedwater in the boilers, some attention must be given to that phase of the subject.

At this point it may be well to define the province of the 'feedwater heater.' It has been claimed that this economical device on boilers in the matter of raising the temperature of the feedwater, will also remove scale from water. This is a fallacy. It is true that the carbonate of lime and magnesia will be precipitated out at the boiling point, but it may be remarked here that the time during which the water remains in the heater is too short to allow of a complete precipitation; and then, again, the temperature usually found in heaters is far short of the boiling point. Calcium sulphate is thrown out of solution at temperatures considerably above the boiling point, so that it is but little acted upon in the heater.

The feedwater heater is a useful device, but its usefulness lies in its power to raise the temperature of the feedwater and not to remove scale.

But to return to the subject of the direct and continuous treatment of feedwater by the use of chemicals. Precipitation may be undertaken when the water contains the carbonates of lime and magnesia or the sulphate of lime; then the precipitates can be removed easily from the boiler by 'blowing off' and washing them out at frequent and regular intervals. At best this method is but very ineffective, for neither by the employment of slime catchers or mud collectors, nor by blowing out or washing the boilers, can the precipitate be wholly removed. In either case a large part of the precipitate will remain behind in the boiler; and all these precipitates bake fast on heating the boiler water. If precipitation in the boiler is carried on it is not a matter of indifference what compound is used. The chief precipitants are caustic lime, caustic soda, and soda ash. If a water containing principally bicarbonates is used, no caustic lime may be used as a precipitant, for by its use the quantity of precipitate in the boiler would be doubled; if the water contains lime sulphate, soda ash should be used; if it contains bicarbonates of lime and magnesia and lime sulphate, caustic soda or soda ash may be used to effect the precipitation.

Perhaps soda ash may be considered the universal precipitant from its general efficiency and cheapness. There are some objections to its use as it 'works' the joints of the steam lines, causing 'foaming' and saponifies the oil used for lubrication in the cylinders, but all these difficulties can be largely overcome by judicious use, and by following rigidly the directions as determined by a thorough and complete chemical examination of the feedwater.

All secret compounds, vegetable or mineral, which act injuriously on the boiler plates or bring further impurities into the boiler are objectionable, and in every case are too dear when soda ash, their chief component, is so cheap. If soda ash is to be used in the direct precipitation of the scale-forming ingredients in the boiler itself, it should be dissolved in water and fed into the feedwater after it has passed through the heater. A stated amount should be used through a given period of time. All these data can be obtained from an analysis of the feedwater and the amount of water evaporated.

The principal minerals occurring in boiler waters and in their precipitants are the following:—

SiO_2 = silica.
$CaSO_4$ = calcium sulphate.
$CaCl_2$ = calcium chloride.
$CaH_2(CO_3)_2$ = calcium bicarbonate.
$MgH_2(CO_3)_2$ = magnesium bicarbonate.
Na_2CO_3 = sodium carbonate (soda ash).
$NaHCO_3$ = sodium bicarbonate.
$NaOH$ = sodium hydroxide (caustic soda).
$Ca(OH)_2$ = calcium hydroxide (caustic lime).
Fe_2O_3 = ferric oxide.
Al_2O_3 = alumina.

Sulphate of lime forms a very dense, hard scale that is difficult to remove. Soda ash changes the sulphate into a carbonate of lime, which is more readily handled. The reaction is as follows:—

$$CaSO_4 + Na_2CO_3 = Na_2SO_4 + CaCO_3$$

Roughly speaking, 1 lb. of calcium sulphate requires ·8 lb. soda ash to effect the transformation. There is no secrecy or legerdemain in this. If the boiler compound does not do this it is a mere farce and makeshift, that is, if the scale is of the above nature. One does not have to reason very far to see that the soda ash must be a constituent of a good boiler compound for such a water.

Perhaps it will be advantageous to illustrate the various types of boiler waters found in the Pittsburg district by copious examples of analyses. All results of water analyses are reported in grains per United States gallon.

The following analysis, No. 1, shows an alkaline water, and complaint was made that it primed in the boilers and also made a deposit upon the feedlines leading from the heater. It is obvious that the only precipitation possible would be from the carbonates of lime and magnesia.

Water No. 1.

	Grains per gal.
Silica	·163
Iron oxide and alumina	·163
Sodium chloride	5·298
Sodium sulphate	3·554
Sodium carbonate	5·234
Calcium carbonate	7·172
Magnesium carbonate	2·392
Organic matter	2·509
Total solids	26·485

An examination of this deposit revealed the following:

	Per cent.
Silica	·83
Iron oxide and alumina	2·16
Lime	48·56
Magnesia	3·04
Combined water	4·17
Carbon dioxide	38·37
Oil	3·28

It is readily seen that the deposit is largely carbonate of lime, which the heater has thrown out of solution but did not retain in itself as it was supposed to do.

It was recommended in connection with the above water to use tannic acid to destroy the alkalinity of the water and to overcome foaming. Resort is had to this chemical in cases of alkaline waters. Also, some manufacturers of secret boiler compounds use tannate of soda to overcome the difficulties occasioned by the use of straight soda ash, that is, priming and cutting the oil from the cylinders.

Water No. 2.

	Grains per gal.
Silica	·968
Iron oxide and alumina	·087
Sodium chloride	·210
Sodium sulphate	3·966
Calcium sulphate	7·833
Calcium carbonate	3·563
Magnesium carbonate	3·703
Organic matter	2·007
Total solids	22·337

In water No. 2 the very troublesome scale-forming ingredient, lime sulphate, makes its appearance. No compound of any kind was used in this water. Naturally one would expect a very hard, dense scale in the boiler, made up of lime sulphate. lime, and magnesia in so far as the heat had driven out the carbon dioxide from their carbonates or water from their hydrates.

The analysis of the scale gave the following :—

	Per cent.
Silica	9·27
Iron oxide and alumina	13·26
Lime	24·98
Magnesia	14·63
Sulphur trioxide...	28·35
Ignition oss	9·88

This scale is about what would be expected. The analysis would indicate that lime is present principally as a sulphate.

The above waters and the resultant scale will serve as general types. The scale is bound to be determined by the nature of the water, unless the feedwater is softened, and the scale transformed by chemicals.

No. 3, Youghiogheny River Water.

	Grains per gal.
Silica	·222
Iron oxide and alumina	·117
Sodium chloride	·163
Sodium sulphate..	·282
Calcium carbonate	nil
Magnesium carbonate	·341
Calcium sulphate	1·008
Organic matter	·586
Total solids	2·719

Analysis No. 3 represents the Youghiogheny River water before its contamination by mine drainage. One would hardly expect to find a water any lower in scale-forming ingredients, yet this water does form scale. The burden of the scale is calcium sulphate. It is always necessary to use a small amount of soda ash per day for every 100 h.p., as note the following illustration :–

1 h.p. evaporates per hour ...	4 gal. water
100 h.p. evaporates per hour ...	400 gal. water
100 h.p. evaporates per day (10 hours)	4,000 gal. water
1 gal. water contains lime sulphate	1 gr.
4,000 gal. water contains lime sulphate	$\frac{4}{7}$ lb.
1 lb. lime sulphate requires ..	8 lb. soda ash
$\frac{4}{7}$ lb. lime sulphate requires ..	$\frac{32}{70}$ lb. soda ash

A 1,000 h.p. plant would require (approximately) 5 lb. soda ash per day.

This amount of soda ash would change the troublesome lime sulphate to a carbonate which can be handled more readily, and at a cost of 5 cents per 1,000 h.p. per day. The same amount of expensive secret boiler compound would cost at least 40 cents per day.

Then, again, to show the rapidity with which scale will accumulate even from a pure water like the above, attention is called to the following calculation based upon a 100 h.p. boiler :—

1 day's run 10 hours, lime sulphate ...	$\frac{4}{7}$ lb.
6 ,, ,, ,, ...	$3\frac{3}{7}$ lb.
25 ,, ,, ,, ...	14 lb.
300 ,, ,, ,, ...	168 lb.

The above figures do not take into consideration the scale that is formed from the carbonates, which would augment considerably this total. And it is needless to state the increased cost for fuel that is brought about by scale in the boilers, and the many other things with which every boiler man is familiar.

Below are given the analyses of two waters that show a marked contrast in their nature. The total mineral is high in each, but the scale-forming ingredients in No. 4 are only about one-seventh of the amount in the No. 5.

	Water No. 4.	Water No. 5.
Silica	1·426	·373
Iron oxide and alumina ...	·093	·367
Sodium chloride... ...	4·561	4·395
Sodium sulphate ...	30·233	—
Sodium carbonate ...	1·914	—
Calcium chloride ...	—	·238
Calcium carbonate ...	9·694	1·804
Calcium sulphate ...	—	71·749
Magnesium carbonate ...	3·912	8·715
Total minerals ..	51·833	87·641

The first contains over 30 gr. to the gallon of the soluble alkaline sulphate Na_2SO_4, while the second contains almost 72 gr. of $CaSO_4$ which forms a dense, hard scale very difficult to remove from boilers.

Another water, No. 6, shows magnesium chloride. This compound is said to be decomposed in the boiler in concentrated solutions, forming hydrochloric acid which attacks the metal work.

	No. 6.	No. 7.	No. 8.	No. 9.
Silicia	·187	·012	·624	·432
Iron oxi le and alumina	·085	—	·128	·087
Alumina	—	·041	—	—
Ferrous sulphate ..	—	·201	—	—
Sodium chloride ...	8·714	1·233	·578	·866
Sodium sulphate ...	—	·392	1·305	2·341
Sodium bicarbonate ...	—	1·226	1·908	8·893
Calcium sulphate ...	12·442	—	—	—
Calcium bicarbonate ...	—	3·729	11·858	11·016
Calcium carbonate ...	2·563	—	—	—
Magnesium carbonate ...	2·295	—	—	—
Magnesium bicarbonate	—	·810	5·493	6·681
Magnesium chloride ..	2 543	—	—	—
Organic matter ...	7·511	·805	—	—
Total solids ...	36·340	8·449	21·894	30·316

Analysis No. 7 shows a pure, clean water from a flowing well, showing the minerals as they exist in their natural form. Along the same lines are shown two more deep-well waters of the bicarbonate type, Nos. 8 and 9.

In boiler practice the bicarbonates are destroyed in the evaporating process, hence it is the usual practice to make all calculations to the straight carbonates. As has been noted, the bicarbonates are destroyed upon boiling in the boiler, and so the water loses a part of its temporary hardness. In a water softener these bicarbonates of lime and magnesia would probably be transformed by milk of lime into other chemical combinations, which will be touched upon more fully under the discussion of the 'water softener.'

The next analyses, 10 and 11, show a very bad water, raw, and after its treatment with soda ash. The consumption of soda ash was large, but only by its use was the plant able to operate during a prolonged dry season in a district where better water was not to be obtained under any circumstances.

	No. 10. Raw Water.	No. 11. Treated Water.
Silica...	1·505	·443
Iron oxide and alumina...	2·333	1·283
Sodium chloride ..	29·637	35·797
Sodium sulphate ...	59·409	121·473
Calcium sulphate ...	56·311	—
Magnesium sulphate ...	24·851	—
Free acid (SO_3) ..	2·725	—
Sodium carbonate ...	—	21·456
Calcium carbonate ...	—	34·579
Magnesium carbonate ...	—	15·711
Total minerals ...	176·771	230·742

Soda has changed sulphate of lime and magnesia into their carbonates, and neutralised the 'free acid' present. An excessive amount of soda ash was used, due to an improvised method of softening. In such a bad water as the above some sort of sedimentation must be allowed. In this case there was a continuous system of rapid filtration which removed some of the precipitate from the lime and magnesia. In other words, there was a continuous water softener of a crude type at work.

These 'improvised water softeners' were seen aplenty during the prolonged drouth of the summer and fall of 1908, especially in the coke regions of the Pittsburg district. Many operators were caught napping, but the plants had to be operated, and there was nothing to be done but to use the water at hand to the best advantage. In many instances the raw mine water was used, the water being softened by the use of soda ash.

The following is so-called 'sulphur water' from mine drainage taken from a mine that has been operating for the past 25 years and shows very strong in 'free' sulphuric acid. An iron or steel pipe is eaten away in a short time by this water, it is so strong in acid.

A part of the 'free acid' in this analysis may be combined with iron oxide and alumina to form sulphate of iron and aluminum, but it is highly improbable that such is the case. It is seen that the sulphuric acid forms quite a strong solution in the water. Every gallon contains 1,266·82 gr. or ·18 lb. (H_2SO_4) sulphuric acid.

	Grains per gal.
Silica	1·516
Iron oxide	45·488
Alumina	116·052
Sodium chloride	57·734
Sodium sulphate	67·564
Calcium sulphate	77·282
Magnesium sulphate	18·286
Free acid (SO_3)	1,013·455
Total minerals ...	1,397·377

Water Softeners.

There are two general types of apparatus for softening water: (*a*) the continuous system by which the water is continually treated and filtered; (*b*) the intermittent system by which the necessary chemicals are injected and the precipitate allowed to subside for a number of hours. The latter seems to be the more satisfactory method, though it involves a greater expenditure, since there must be a number of settling tanks. The water softener is the only logical way to handle a bad water. By it all the precipitates are removed from the water before it enters the boiler. In the direct-precipitation method, which has been described before, all the precipitates are thrown down in the boiler and must be washed or blown out. This could never be accomplished with a really bad water.

Adhering to the original precipitants of caustic soda, caustic lime, and soda ash, as they are the cheapest known by far, the general ideas of the softening process will be shown. It is understood that other precipitants, as tri-sodium phosphate, barium compounds, etc., may be employed, but this makes the process too expensive. These expensive precipitants are usually to be found in secret boiler compounds. Some contain organic chemical compounds, as sugar, etc.

As has been noted, lime sulphate is a very bad scale-forming ingredient. It is transformed by soda ash, as is the calcium chloride, after the following chemical reaction:—

$$CaSO_4 + Na_2CO_3 = Na_2SO_4 + CaCO_3$$
$$CaCl_2 + Na_2CO_3 = CaCO_3 + 2NaCl$$

The bicarbonates of lime and magnesia are handled by caustic soda after the following reactions:—

$$CaH_2(CO_3)_2 + 2NaOH = CaCo_3 + Na_2CO_3 + 2H_2O$$
$$MgH_2(CO_3)_2 + 4NaOH = Mg(OH)_2 + 2Na_2CO_3 + 2H_2O$$

These same bicarbonates are also transformed by caustic lime as follows:—

$$CaH_2(CO_3)_2 + Ca(OH)_2 = 2CaCO_3 + 2H_2O$$
$$MgH_2(CO_3)_2 + 2Ca(OH)_2 = Mg(OH)_2 + 2CaCO_3 + 2H_2O$$

Caustic lime and soda ash are the staple precipitants for the waters usually found in the Pittsburg district. In short, the lime takes care of the bicarbonates, and the soda ash the sulphates in a water softener. There is nothing better for the money.

In case of an alkaline water, if it is very strongly alkaline, enough to cause foaming in the boilers, it can be treated by the use of caustic lime with either magnesium sulphate, iron sulphate, calcium sulphate, or sodium bisulphate, according to well-known chemical reactions.

In conclusion: It has been the aim of this article to show the fallacy of treating boiler waters other than by logical chemicals and to discourage the use of expensive compounds, when soda ash alone is most efficacious for the direct precipitation in the boilers; to encourage the substitution of water softeners for this direct precipitation, as it will more than pay in the end when the water is very heavy in scale-forming ingredients and large amounts of water are evaporated daily; to insist upon the coal and coke operator getting into line with his manufacturing brother and being up to date in the matter of a careful and systematic examination and treatment of his boiler feedwater, for every dollar so expended will yield profitable returns in increased efficiency of the boilers and in decreased cost of labour and repairs."—*Mines and Minerals*, Feb., 1909. (A. R.)

METALLURGY.

CUSHIONING VIBRATIONS OF CAM-SHAFTS. "The vibrations in a stamp-mill are caused by the impact of the falling stamp on the ore between the die and the shoe, and those set up in the cam-shaft when the cams strike the tappets in the act of raising the stamps. The vibrations due to the falling stamps are cushioned to a certain extent by the layer of ore on the die, and are confined to the stem and mortar-block, and are necessary in this type of machine. The vibrations of the cam-shaft are very disagreeable. They cause nuts to loosen, bolts to break, cam-shafts to crystallise, cam-shaft bearings to wear out, and so occasion a loss of power. No attempt has thus far been made to cushion these vibrations.

The drawings* represent a design I have made to lessen these vibrations, and, incidentally, to afford the mill-man a means of alternating the drop of the stamps without touching the tappets. The proper material for cushions, which will be 8 in. square, is yet a subject for experiment. Pieces of thick belting may serve well. When it is required to lower the stamps, pieces of this material may be taken out and replaced at will. An opening is provided in the front of the chair for this purpose. The dap or gain in the battery post on which the chair sits may be cut in an inch or two and filled with some cushioning material for the cam-shaft bearing, which fits in the chair, but does not touch the chair except on the sides, the bottom of the chair being open. The collars on the shaft should be of sufficient diameter to always bear on the sides of the chair. The lip on the bearing, which may be called the plug, will keep the grease where it will fall into the receptacle on the chair, or cotton waste may be used to catch it. The plug is shown solid, but this is not necessary. Manufacturers of stamp-mills should do something to alleviate the miseries of the cam-shaft floor, and these drawings are submitted in hope that the subject may be taken up."—ALGERNON DEL MAR.—*Mining and Scientific Press*, Dec. 26, 1908, p. 877. (W. R. D.)

MACQUISTEN PROCESS OF FLOTATION.—"Mr. A. Selwyn-Brown's short review of modern development on the metallurgy of lead and zinc is most timely. After disposing briefly of the Huntington and Herberlein, the Carmichael-Bradford, and the Savelsberg processes of wasting and agglomeration, Mr. Selwyn-Brown touches on briquetting concentrates and fine sulphides, sintering, and so on.

But the most important section of the paper is that descriptive of the Macquisten process of flotation. the recent invention of a Glasgow investigator. The Macquisten process differs from the Potter, Delprat, De Bavay, and Elmore processes in employing no chemicals. The sulphide particles are separated from the gangue by utilising the surface tension of water by means of a simple mechanical device. The operation is based upon the fact that sulphide minerals are positively affected by the surface tension of water while the rock constituents are not so affected. 'The surface tension of the water spreads around each sulphide particle like an envelope, leaving, probably, between the surfaces of the water and the mineral, a bubble of air large enough to float the sulphide. No envelope is formed around silicates. The water appears to penetrate them and causes them to sink.'

The process is conducted in a concentrator composed of a simple cast iron tube, 6 ft. in length and 1 ft. in diameter. Near each end are two tires resting on supporting rollers to enable the tubes to be revolved with little friction. Inside the tubes is a helical groove with an inch and a half pitch running from end to end, as in a boltnut.

'The tube is revolved at the rate of about 30 revolutions per minute. The pulp is screwed through the tube and is so guided by the helical screw groove as to gain the mineral particles repeated opportunities to glide upon the surface of the water. Good concentration can be made in one tube. The inventor's experiments showed, however, that when four tubes were joined in series so that one fed into the next one following, some of the finest particles did not get properly started on the water's surface until they had nearly reached the discharging end of the fourth tube.

This process has been tried on a larger scale at the Adelaide Mill, Nevada. It was run with an ore composed of chalcopyrite, pyrrhotite, and pyrite with a little blende and galena which previous plants had failed to handle profitably. Each tube successfully treated from 5 to 6 tons of ore per 24 hours, concentrating in the ratio of 11 to 1.'"—A. SELWYN-BROWN, *The Engineering Magazine*, Sept., 1908.—*Canadian Mining Journal*, Oct. 15, 1908, p. 536. (C. B. K.)

SETTLING SLIMES IN CYANIDE TREATMENT.—"The author describes a process which has given remarkable results, both in the completeness of the separation effected and in the small proportion of liquid carried off by the solid matter, which is removed as it reaches the bottom of the tank. This is of inverted pyramidal shape, and connects at the bottom through an 8-in. square aperture with a closed box in which a 10-in. belt is made to travel slowly. The method may be applied in practice in either of two ways. First, as an intermittent process in which the belt discharge of solids would be delivered into a second similar tank where a weak solution or wash water could be sprayed on to the belt to loosen and disintegrate the slime; or a mixer may effect this purpose. After removal of the required proportion of solids the solution would be withdrawn to the original level and a second charge introduced. Secondly, the introduction of a suction filter near to the surface of the liquid in the separator, permits of a continuous process being carried on. The percentage of moisture in the discharge is inversely as the proportion of solids in the separator charge, and the first portions of a discharge from any given separator charge are always better than the succeeding portions, hence the density of the separator charge is maintained at its initial figure, the moisture in the discharge would be kept at a minimum and the rate of discharge increased. By an automatic periodical cutting off of the suction and application of small back pressure the filter screen is kept clear from caked slime and clear liquid is continually drawn from a pulp of sp. gr. 1·4. The belt and suction must be so correlated that the solids discharged are proportionate to the liquid sucked out, and the discharge from a pulp of the density stated will uniformly contain about only 22·5% of moisture. If there be two suction pipes with screens they are kept clear by back pressure being applied alternately to each. The continued use of screen suction would not be possible unless the solids were being extracted by the conveyor belt. The procedure in the wash settlers is practically the same. The solids are washed off the belt by precipitated solution and water, and the suctions withdraw an equal amount of liquid, which is passed through the extractor boxes and returned from the sump. In other processes completeness of washing is dependent on time and capacity of plant, as each instalment of slime is taken separately and washed solution passed through it as long as economically possible, when it must be removed, whether well washed or not, to make room for the next instalment; but in this method continuity is unimpeded and slimes may be washed out with any reasonable quantity of weak solution without affecting the time of treatment—the only factor affected being the capacity of the extractor boxes.

---

* These show a chair holding some cushioning material such as layers of belting, on which the bearing is carried.

The advantages of this method are: (1) Absence of pumps, carrying machinery and complicated appliances; (2) the separation of sands from slime is not called for, and thus disadvantages attendant on dealing with such classes of material as depend on close classification are obviated; (3) it is possible to wash to the best advantage and thus reduce the loss by residual moisture; (4) the degree of fineness or uniformity is immaterial; (5) the initial cost is very low, and niceties of adjustment are not requisite; (6) the only limit to the thickness of a charge is the point at which it will not run, and best results are obtained from charges with high percentages of solids; (7) the process may be continuous. The paper is accompanied by diagrams and tables showing the arrangement and the results of working."—*Electrochemical and Metallurgical Journal.—Canadian Mining Journal*, Oct. 15, 1908, p. 534. (C. B. K.)

New High Speed Steel—Novo Superior.—"The casual announcement made by Prof. Arnold, of Sheffield University, in his Royal Institution lecture, concerning a new high-speed steel, has attracted considerable attention in the steel world. Messrs. Jonas and Colver (Limited), of Sheffield, the makers, are still keeping the composition of the new steel secret. Prof. Arnold claimed for it quadruple the cutting power of all existing high-speed steels, and he has since expressed the opinion that in a comparatively short time all the high-speed steel manufacturers in the country will be making the new material. The ordinary high-speed steel, 'Novo,' has been well known for some years, and the new material has been named 'Novo Superior.' A remarkable fact is that it is a water-quenched steel, and it is the first high-speed steel produced that can be hardened in cold water without danger. Water-quenched steel is usually associated with brittleness, and with the quality of 'letting down' at a temperature far below a red heat. The new steel, however, will not crack at high temperature when plunged into water. It can also be hardened, with equal ease, by oil or even in a cold blast, these being the methods of hardening adopted in the case of the old high-speed steel.

The advantage claimed for 'Novo Superior' is that it will go on working at high temperature for a longer period than existing high-speed steels. Tools made from it will not require regrinding nearly so often as those made from the present high-speed steels, nor will they wear at anything like as great a rate. Prof. Arnold has given some interesting comparative figures of temperatures. Ordinary carbon steel, he said, 'lets down' at a temperature of about 250° centigrade—a low black heat—while ordinary high-speed steel will go up to 600°; and he estimates the working capacity of the new steel at 700° centigrade. Prof. Arnold and Mr. Winder agree that the new steel does not mark by any means the final development of high-speed steels."—*Times Engineering Supplement*, Feb. 10, 1909. (J. A. W.)

Coke Analysis.—"An investigation of the great discrepancy which existed between laboratory tests and actual practice in the coke output of bituminous coals was carried out by F. W. Hinrichsen and S. Taczak at the Königliche Materialprüfüngsamt, Gross Lichterfelde. The following new method of proximate analysis was determined upon: Four to 5 gm. of finely pulverised coal are placed in a Rose crucible of 40 c.c. to 50 c.c. capacity; diameter at bottom, 20 mm.; height, 50 mm.; top diameter, 50 mm. Hydrogen is introduced through the tube for 10 minutes, then it is heated for two minutes with moderate bottom flame, then for five minutes with bottom flame and two side flames, or the heating is continued until the yellow flame disappears, using the three burners. It is cooled in the current of hydrogen. Determinations made by this method usually show several per cent. higher coke output than the older methods and agree very well with practice. In a set of comparative determinations made with the two "Muck" methods and this one, the results derived in practice were a shade lower than the experimental determination in hydrogen, and several per cent. nearer than the "Muck" determinations."—Hinrichsin and Taczak.—*Electrochemical and Metallurgical Industry*, Jan., 1909, p. 34. (A. McA. J.)

MINING.

Electrically Driven Mine Pumps.—"*The Series-wound Motor for Mine Pumps.*—In the early days of the electrically driven mine pump, the series-wound motor was universally employed, partly because the shunt-wound motor was not then properly understood, and partly because alternating currents were not then used at mines. The method employed at that time in England, and I believe, very largely at the present day, was, and is, to run the pump for a certain number of hours during the day, lowering the water that had run into the workings sufficiently to allow work to be continued until the next pumping period. One of the difficulties was starting the pump. It usually had to be started against the resistance of the inert column of water in the pipes leading to the bottom of the shaft, and sometimes in the entire column up to the surface. Considerable torque was required at the moment of starting, and the series-wound motor furnished that quite easily. Unfortunately, however, starting was only a part of the work that the motor was required to do, and starting against the inert column of water was only one of the conditions under which the pump was required to work. It was not unusual for the pump to be started and left to run by itself for a certain time, an attendant going to it occasionally, or it even being switched off at surface. This would have been all right if the quantity of water to be pumped had been the same day by day, but in a great many cases at least in Great Britain, especially at shallow mines, the quantity of water varies with the rainfall on the surface, the water making its way into the workings, not immediately after the rainfall, but at some later time depending upon the overlying strata. This in the case of the series-wound motor led to trouble, for when the pump began to lose water, the series-wound motor would run away, and unless the current was promptly switched off, often injured itself, or the pump, but more particularly itself. This is one of the great defects of the series-wound motor. It changes its speed so much with a varying load. In coal cutting for instance, when a nodule of pyrites is met with, the series-wound motor will slow up, and, providing there is power behind it, will cut its way through, but immediately the nodule is passed, it will commence to race unless the attendant is on the lookout, and is able to switch off the current, or stop it in other ways.

At the present time, there is apparatus on the market, that would enable the series-wound motor to be employed for pumping with perfect safety and without the necessity of constant attendance. Automatic appliances are arranged to start a motor when the level of the water rises to a certain point, and to stop it when the level has fallen to a certain point. The automatic appliance consists of a solenoid electromagnet with a tripping contact, actuated either directly or indirectly by a float. The solenoid electro-magnet then actuates the ordinary starting resistance.

Shunt-wound and Three-phase Motors for Mine Pumps.—The shunt-wound motor is undoubtedly the best all-round motor for general work about a mine, including pumping, and the three-phase motor is quite as good, except for the one objection that while the speed of the shunt-wound motor can be increased from one to six times, regulation of the speed of the three-phase motor is difficult. Both the shunt-wound continuous motor and the three-phase motor are self-governing, and both run at nearly constant speed—within about 4%.

Where it can be arranged so that the pumping is done for a certain number of hours during the day, either the shunt-wound, continuous motor with continuous current, or the three-phase motor with alternating current will answer perfectly. They will deal easily with any variations of load incidental to the ordinary working of the pump. So long as the water that is made during a certain portion of the 24 hours can be pumped out so as to leave the workings free during working hours, the shunt motor, or the three-phase motor, running during those hours at constant speed, will do the work. Also with varying quantities of water, if varying times can be applied to getting the water out, again the shunt-wound and three-phase motor working at constant speed will do the work. But where large increases in the amount of water must be pumped at certain seasons of the year, after heavy rains, or due to other causes, and increased time cannot be given to getting the water out, the pumps must be run at a higher speed.

The shunt-wound motor lends itself readily to increased speed. By adding a resistance to the field coils of the motor, which is thrown into the field circuit at the time that a higher speed is required, the speed may be increased as much as 20%, and even in certain cases 25%. But with the ordinary construction of the continuous-current motor, as the speed increases, and particularly as it approaches 20% to 25% above the rated speed, sparking commences and becomes somewhat violent at the brushes. This difficulty is easily overcome, if it is known, when the motor is ordered, that a varying speed will be required, for motors are now made with what are called commutating or auxiliary magnet poles, placed between the ordinary field magnet poles. These auxiliary poles suppress the sparking which takes place when the speed is greatly increased. There is not space here to explain the full action of the apparatus, but it may be mentioned that the increased sparking is due to the weakening of the magnetic field, produced by the insertion of the resistance in the circuit of the field coils. As the resistance added to the field coils is increased, the magnetic field produced by the field magnets is decreased, and the speed of the motor may be proportionately increased. By the aid of the commutating poles, the speed may be increased up to as much as six times the normal. Probably the ordinary motor, with resistance added, would answer the purpose, because the pump could hardly be allowed to run at more than 20% or 25% above its normal speed.

In the case of the three-phase motor, a similar method of regulation is not available because the speed of alternating-current motors depends, not upon the current strength, but upon the number of alternations of the current. A certain variation in speed can be obtained, say, half speed, three-quarter speed, if it is known beforehand that it will be required, by altering the conditions of the coils of the stationary part of the motor, but no intermediate speeds can be obtained, except by the use of wasteful resistances.

Centrifugal Mine Pumps.—It may perhaps be noted, that while the three-throw plunger pump has been such a great favourite, and such a good servant, the centrifugal pump now has many advocates. Not very long ago the centrifugal pump could only be employed for low heads, but now single centrifugal pumps are made for heads of 100 ft., and by placing two or more pumps in series, any head that may be desired can be worked against. I understand that the Worthington company has supplied centrifugal pumps raising water as high as 2,000 ft. in a single lift. The centrifugal pump is much more convenient than the three-throw plunger pump. It occupies less space for it can be connected directly to the axle of the driving motor, and the later designs of centrifugal pumps are certainly more efficient than the older forms of the three-throw pump. It also, however, has certain features that have to be reckoned with, when applying pumps. The centrifugal pump can be designed for constant quantity, constant speed, or constant head, but when the conditions for which it was designed are changed much, the efficiency of the pump decreases rapidly. Thus, in the case of the pump designed for constant quantity, its highest efficiency is when it is delivering that quantity. If the quantity is increased, the efficiency commences to fall, and with a comparatively small increase, falls very rapidly. On the other hand, if it is not delivering that quantity, its efficiency is also very much less. I have the results of a test of a pump designed to deliver 1,250 gallons per minute against a head of from 100 ft. to 117 ft., the pump to run at 700 r.p.m. At 700 r.p.m., the efficiency of this pump was about 72%; at 750 r.p.m., it was still 72%; at 800 r.p.m. it had fallen to 70%; at 900 r.p.m. it was only 58% and at 950 r.p.m. it was only 40%. Similar conditions are met with in pumps designed for constant speed or constant head. So long as centrifugal pumps are working within a short margin of the condition for which they are designed, they work at high efficiency, but when those conditions are departed from the efficiency falls very rapidly. These facts, it appears to me will bear upon the question of centrifugal pumps for mines.

The best form of pump would probably be that designed for constant speed, provided that the quantity of water to be dealt with did not vary much. In those cases where the water might vary, probably the pump designed for constant head would be the best. With constant speed, the pump may be driven by either the shunt-wound motor, or the three-phase motor, and should give at least as good results as the three-throw pump, but if the speed has to be altered the shunt-wound motor is practically the only one available. With reference to the matter of the starting torque, in which the shunt-wound motor is comparatively weak, this difficulty may be overcome by having a few turns of wire on the motor, making it temporarily a compound-wound motor, a few turns being in series with the armature. These few turns will give the starting torque required, and can be cut out after the pump is running at full speed. It should be mentioned, that the improved efficiency of the centrifugal pump has produced the natural result, improved efficiency in the three-throw pump."—SYDNEY F. WALKER. — *Engineering and Mining Journal*, Feb. 20, 1909, p. 422. (A. R.)

MISCELLANEOUS.

WHAT IS A MINERAL?—"Dr. Gregory, Professor of Geology of Glasgow University, read a paper before the Mining Institute of Scotland, held in Glasgow, bearing on a subject which has conduced to much troublesome litigation of late—namely, 'What is a Mineral?' The definition of the term 'mineral,' was a question of practical importance in the interpretation of leases and Acts of Parliament. The minerals were often owned by a different proprietor from the land. Thus a railway company did not buy the minerals 'under the land' required for its lines, It had the right to support from the minerals without payment until their owner could mine them. This arrangement was intended to protect the mineral industry, while it saved the railway companies, at the time of the construction of a line, having to purchase these minerals of no value, for which it might never have to pay. Therefore the less that was included under the term 'mineral,' the less a railway company might be called upon to pay in addition to the original price of the land. The term 'mineral,' proceeded Prof. Gregory, was undoubtedly long used to mean all materials that were not animal or vegetable. It was not until 1850 that Dana, the great American mineralogist, made the first attempt to restrict the term, and he proposed to limit it to what had previously been called 'simple minerals' or 'mineral species.' He proposed to re-christen the mineral kingdom the inorganic kingdom, and limit the term 'mineral' to one section of inorganic substances. He proposed this restricted use of the word 'mineral' merely as a convenient convention in the science of mineralogy, and in his more popular writings continued to use the term 'mineral' in its wider sense. In recent years the attempt had been made to adopt the term 'mineral' as a synonym for mineral species, and there was a widespread idea that this use of the word 'mineral' had been accepted by British mineralogists. This view was, however, incorrect. The leading British authorities on mineralogy, both in 1850 and at the present time, had rejected Dana's proposal, and applied the term 'mineral' to all inorganic constituents of the earth's crust The term was thus used in Mier's Mineralogy, the standard British text-book, and by the Mineralogical Department of the British Museum and the Geological Survey. The restriction of the term 'mineral' to mineral species was against established usage and against common sense. Indeed, if it were so restricted, coal, most ironstone, most ores, mineral oil, limestone, etc., were not minerals. Rocks, although often made up of minerals, would not be mineral, and the only minerals of commercial value would be a few scarce crystals and, doubtfully, some occasional masses of iron ore. The term 'mineral' was sometimes used in leases, to include less than the whole mineral kingdom. In such cases the term did not mean a mineral substance or a 'mineral species,' but a mineral of commercial value. A commercial mineral was any constituent of the earth's crust which had a value of its own, apart from its value as a soil or as a support. The distinction between mineral substances and mineral species was quite indefinite, and no sharp line could be drawn between them; and the definition of minerals in contrast to mineral species as the constituents of the earth's crust was both historically and scientifically correct." — PROF. GREGORY. — *London Mining Journal*, Feb. 20, 1909, p. 245. (A. R.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

THE MECHANICAL APPLIANCES OF THE CHEMICAL AND METALLURGICAL INDUSTRIES. A complete description of the machines and apparatus used in chemical and metallurgical processes for chemists, metallurgists, engineers, manufacturers, superintendents, and students. With 292 illustrations. By OSCAR NAGEL, Ph.D. 8s. 6d. (New York, U.S.A.: The Author. London: *The Mining Journal.*)

"Although one may not be able to accept the author's modest statement in the preface that he has said 'everything that is of interest to the chemical and metallurgical engineer regarding superheated steam, turbines, and especially gas power,' there is no doubt that this useful work will be found of great value to both classes and to other kinds of engineers.

Written by a German and in characteristic German style, but essentially an American book for the Americans, it touches upon the more important general matters with which the chemical and metallurgical engineer are connected, and deals at some length with many of them.

Although the unfortunate English inventors, discoverers, and manufacturers have received but scant recognition, their pioneer and development work have been fully exploited, and a paper by Oscar Guttmann, read in England, on 'The Works Chemist as Engineer,' has been reproduced as an appendix.

To give a complete list of the many branches of engineering which are dealt with, would be impossible, but such matters as the transportation of solids, liquids, and gases, the production and use of producer gas, the firing and stoking of furnaces, and the use of electric power, are, perhaps, most interestingly considered.

The work may be strongly recommended to those who require a short and pithy book of reference reasonably up to date, and to those who wish to see placed before them briefly but succinctly, the principal lines on which recent developments have proceeded. The principal types of machine and apparatus in use in America are dealt with fairly fully and without more advertisement for manufacturers than can be avoided when, as is necessary in the case of such a work as this, it is necessary to draw upon so many trade catalogues to provide the illustrations." —*The Mining Journal* (London) Literary Supplement, March 27, 1909. (W. A. C.)

Report on the Mining and Metallurgical Industries of Canada, 1907-8. Containing numerous photographic illustrations, maps, plans, and tables. (Ottawa: Government Printing Bureau, 1908.) 9¾ × 6½, xvi. and 972+35 pp. $1.

"The most conspicuous mineral products are coal, copper, gold, silver, nickel, cobalt, asbestos, and mica. In the case of the last three minerals the Canadian output practically controls the markets of the world—in fact, one variety of industrial mica is only produced in Canada. It is stated that this variety, *phlogopite*, is found only in rocks of the Laurentian age, and its production is restricted to a single region north of Ottawa. Various minerals are associated with the mica deposits, pitchblend being one of them. The volume contains a considerable amount of useful information about the various deposits relating to the history of the discovery, the mode of occurrence, the methods of working, the preparation for the market, and in some cases details of the cost of production. A schedule of the more significant proprietors is also given. The matter is arranged under the heads of the different provinces, extending from Nova Scotia and New Brunswick in the east to the Yukon and British Columbia in the west. Although mineral deposits over this immense range of territory are included in the report, yet there are vast intervening areas still unexplored and untouched. As might be expected, the mining and metallurgical processes employed vary very considerably according to the conditions prevailing in the different neighbourhoods, and while some appear to be primitive, others display in a marked manner an ample appreciation of the best modern practice.

The electro-thermic process does not seem to have made as much progress as might have been anticipated in face of the Government encouragement. There is a frontispiece showing a quantity of pig iron produced electro-thermically, but there are not many references to it in the body of the volume. This may be due to the want of a subject-matter index.

Those in search of information on the subjects treated will be fully repaid for the trouble incurred in consulting the volume." — *Times Engineering Supplement.* (J. A. W.)

Annals of the Cape Observatory. Vol. 10. Spectroscopic Researches; Part 3. A Spectrographic Determination of the Constant of Aberration and of the Solar Parallax. *Wyman and Sons.* Net 2s. 6d.

Boycott, G. W. M. Compressed Air Work and Diving. A Handbook for Engineers. Illustrated. Roy. 8vo. pp. 18. *Lockwood.* Net 10s. 6d.

Clayton, W., and Craig, J. Questions and Answers about Electrical Apparatus. Illustrating General Points and Information concerning Testing Operations, Troubles and Defects of Electrical Apparatus. Engravings. 12mo., limp leather, pp. 176. *W. Wesley.* Net 5s.

Gill, A. H. A Short Handbook of Oil Analysis. 5th edition. Cr. 8vo. *Lippincott.* Net 7s. 6d.

Guttmann, Oscar. The Manufacture of Explosives. Twenty Years' Progress. 8vo. pp. 84. *Whittaker.* Net 3s.

Handbook to the Technical and Art Schools and Colleges of the United Kingdom. Compiled from Official information. With an Index to Course of Instruction. 1st edition. Cr. 8vo. pp. xi-140. *Scott, Greenwood.* Net 3s. 6d,

Mellor, J. W. Higher Mathematics for Students of Chemistry and Physics. 8vo. pp. 664. *Longmans.* Net 15s.

Parry, L. Systematic Treatment of Metalliferous Waste. 8vo. *London Mining Journal.* Net 5s.

Poynting, J. H., and Thomson, Sir J. J. A Text Book of Physics: Properties of Matter. 5th edition; carefully revised. 8vo. pp. 236. *C. Griffin.* 10s. 6d.

Redgrove, H. Stanley. On the Calculation of Thermo-Chemical Constants. 8vo. pp. 102. *Arnold.* Net 6s.

Reed's Engineer's Handbook to the Board of Trade Examinations, etc. 19th edition. 8vo. pp. 794. *Simpkin.* Key, net 7s. 6d.; net 14s.

Rogers, Allen. Laboratory Guide of Industrial Chemistry. 8vo. pp. 170. *Constable.* Net 6s.

Royal Dublin Society. Scientific Proceedings of Vol. II. N.S. No. 31—On the Quantitative Spark-Spectra of Titanium, Uranium and Vanadium. No. 32—On the Spectrographic Analysis of a Specimen of Commercial Thallium. By James H. Pollock. Plates. Cr. 8vo. *Williams and Norgate.* Net each 6d.

Snow, Wm. Benham. Therapeutics of Radiant Light and Heat and Convective Heat. Illustrated. 8vo. *Siegle, Hill and Co.* Net 10s. 6d.

Soddy, Frederick. The Interpretation of Radium. Being the Substance of Six Lectures. Illustrated. (Progressive Science Series.) 8vo. pp. 274 *J. Murray.* Net 6s.

Steinmetz, C. P. Theory and Calculation Transient Electric Phenomena and Oscillations 8vo. *Spon.* Net 21s.

Stewart. R. Wallace. An Elementary Text-Book of Physics. Part 3. Light. Cr. 8vo. pp. 228. *C. Griffin.* Net 36s.

Watteyne, V., Meissner, C., and Desborough, A. The Prevention of Mine Explosions: Report and Recommendations. (U.S. Geological Survey Bulletin.) 8vo. pp. 11. *W. Wesley.* Net 1s.

Selected Transvaal Patent Applications.

Relating to Chemistry, Metallurgy and Mining.

Compiled by C. H. M. Kisch, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(C.) 137/09. Herbert Albert Sandor. Improvements in driving mechanisms. 20.3.09.

(P.) 138/09. James Miners Holman (1), John Leonard Holman (2). Improvements in means for allaying the dust produced in mines and the like, by blasting and other operations. 22.3.09.

(P.) 139/09. Ernest Albert Brinkmann. Sample grading machine. 22.3.09.

(P.) 141/09. John Coutts Shaw. Improvements appertaining to stoppers for bottles and similar vessels. 23.3.09.

(P.) 142/09. John Jones. Water heating apparatus for household stoves. 23.3.09.

(P.) 144/09. Frederick Hudson Shepherd Shepherd (1), Christian August Lagesen (2). Improvements in means for reducing the loss of heat from steam generators, steam pipes and the like. 24.3.09.

(P.) 145/09. George Henry Sugden Martindale. Improvements in movable covers. 24.3.09.

(P.) 146/09. Bruce Edward Tennent. Improvements in amalgamators. 25.3.09.

(P.) 147/09. Frederick Hudson Shepherd Shepherd (1), Christian August Lagesen (2). Improvement in means for heating by electricity. 25.3.09.

(C.) 148/09. Sandycroft Foundry Co., Ltd. (1), James McKinlay (2). Improvements in the manufacture of mortar boxes for stamp mills. 26.3.09.

(P.) 150/09. Frederick Hudson Shepherd Shepherd (1), Christian August Lagesen (2). Improvements in the conversion of natural light into heat, and the utilisation of same for useful purposes. 26.3.09.

(P.) 151/09. Otto Schonherr (1), Johannes Hessberger (2). Improvements in means for production of long electric arcs, more especially intended for the manufacture of compounds containing nitrogen. 26.3.09.

(C.) 152/09. William Ginns Bambridge. Improvements in or relating to flexible shafts. 26.3.09.

(C.) 153/09. Wulf Granat. Improvements in the process of extracting gold by amalgamation, and apparatus therefor. 26.3.09.

(P.) 154/09. Samuel Russell. Improvements in stamp guides. 26.3.09.

(P.) 155/09. George Arthur Webb. Improvements in and relating to jockeys for mechanical haulage. 27.3.09.

(P.) 156/09. Alfred Leonard Parker (1), John Thomas Webster (2). Improvements relating to the utilisation of hydraulic power in turbine machinery. 1.4.09.

(C.) 157/09. John Davis (1). Peter Miller (2). Improvements in drill heating furnaces. 2.4.09.

(C.) 158/09. John Hutchings. Improvements in and relating to internal combustion motor engine turbines or reaction wheels. 2.4.09.

(C.) 159/09. John Hutchings. Improvements in apparatus for supplying combustible fluid under constant pressure. 2.4.09.

(C.) 160/09 William Leonard Holms (1), Leonard Melvil Green (2), John Shield Pattinson (3). Improvements in means for separating water or solution from liquids. 2.4.09.

(C.) 161/09. Anthony Maurice Robeson. Improvements in means for crushing or grinding rocks, ores or the like substances. 2.4.09.

(C.) 162/09. The Honourable Charles Algernon Parsons. Improvements in controlling means for fluid operated motors. 2.4.09.

(C.) 163/09. James Miners Holman (1), Mary Elizabeth Carthew Holman (2), widow of John Henry Holman, deceased, George Bennett Nancarrow and Charles Vivian Thomas, the legal representatives (joint executors). Improvements in stamp mills for crushing ore and other substances. 2.4.09.

(P.) 165/09. George Leonard Webb. Improvements in automatic sampling devices applicable for use with stamp batteries, shaking tables, tube mills or the like. 3.4.09.

(P.) 166/09. Colin Armstrong Ferguson (1), Sidney Charles Herbert Herbert (2). Improvements in protective covers for amalgamating plates. 3.4.09.

(P.) 167/09. Alexander John Arbuckle. Improvements in means for introducing and distributing mixtures of solids and liquids in settling or other vats or vessels. 5.4.09.

(P.) 168/09. Herbert Carter. An improved safety clutch for mine cages or other hauling apparatus used in vertical shafts. 6.4.09.

(P.) 169/09. William Beaver. Device for changing screens of stamp mortar boxes while stamps are in motion. 7.4.09.

(P.) 170/09. Pietro Antonio Chiappero. Improvements in apparatus for actuating shaking conveyors. 7.4.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

ALSTON, R. A. *l/o* Germiston : 44, Margaret Street, Greenock, Scotland.

BELL, H. C. F., *l/o* Cala ; French Rand G. M. Co., Ltd., P. O. Box 25, Luipaardsvlei.

BENNETT, W. F., *l/o* Knights Deep ; New Rietfontein Estates, Ltd., P. O. Rietfontein Mines.

BLACKIE, JNO., *l/o* Bulawayo ; Bwana Mkubwa Mine, Ndola, N.W. Rhodesia.

BLOMFIELD, A. L., *l/o* Day Dawn ; Golden Cycle Mining Co., Colorado Springs, Colo., U.S.A.

BOER, E. C., *to* Rainbow Reef, Colleen Bawn Post Office, Rhodesia.

CARDWELL, R. H., *l/o* Cleveland ; Lizant Rectory, Launceston, Cornwall.

GRAHAM, W. H., *l/o* Mazoe ; Colleen Bawn Mine, Gwanda, Rhodesia.

GIBSON, JAMES, *l/o* Durban ; P. O. Box 247, Johannesburg.

HAMILTON, W., *to* Creaton, Pinjaddah, Western Australia.

HILL, J. WHITELAW, Whitelaw, 26, Newark Drive, Pollokshields, Glasgow, Scotland.

HILNER, E. G., *l/o* Germiston ; P. O. Box 1145, Johannesburg.

KERR, THOS., *l/o* Cleveland ; Benoni Consolidated Gold Mines, Ltd., Benoni.

LEA, J., *l/o* Johannesburg ; 8, Old Jewry, London, E.C.

MACQUEEN, W. P. O., *l/o* Johannesburg : c/o Messrs. Gillanders, Arbuthnot & Co., P. O. Box 174, Calcutta, India.

MANNERS, C. E., *l/o* East Rand : West Rand Consolidated Mines, Ltd., P. O. Box 38, Krugersdorp.

OVENDALE, R., *l/o* Johannesburg; c/o A. S. Mitchell, Paardekraal, Bloemfontein District, O.R.C.

ROSE, A. F., *l/o* New York : Fanti Consolidated Mines, Tarkwa, West Africa.

RUSDEN, HY., *l/o* Johannesburg : (until September, 1909), Glyn Garth, Falmouth, Cornwall.

TAYLOR, J. A., *l/o* Germiston ; Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg.

TAYLOR, WILFRID, *l/o* Denver : Three Sisters Mine, via Barberton.

THOMAS, D. L., *l/o* Lydenburg : P. O. Box 168, Johannesburg.

WELHAVEN, ALF., *to* Oriental Consolidated Mining Co., Tabowie, Unsan, Korea.

WRIGHT, A. L., *l/o* Lydenburg ; Vogelstruis Consolidated Deep, Ltd., P. O. Box 3, Florida.

THE JOURNAL

OF THE

Chemical, Metallurgical and Mining Society

OF SOUTH AFRICA.

The Society, as a body, is not responsible for the statements and opinions advanced in any of its publications.

Reproduction from this Journal is only allowed with full acknowledgment of the source.

VOL. IX. JUNE, 1909. No. 12.

Proceedings
AT
Annual General Meeting,
June 19, 1909.

The Annual General Meeting of the Society was held in the Lecture Theatre of the Transvaal University College, on Saturday evening, June 19th, Mr. R G. Bevington (President), in the chair. There were also present:—

56 Members: Messrs. A. McA. Johnston, E. H. Croghan, W. R. Dowling, C. B. Saner, G. O. Smart, Prof. G. H. Stanley, H. A. White, Prof. J. A. Wilkinson, W. A. Caldecott, E. H. Johnson, Prof. J. Yates, H. A. Adams, D. J. Arkell, A. Avent, W. Beaver, W. Broom, A. A. Coaton, M. H. Coombe, C. A. Corder, L. Evans, J. F. Ferguson, E. A. Furner, A. D. Gilmour, J. Gray, S. A. Herbert, A. Heymann, C. B. Hilliard, J. I'Ons, G. A. Lawson, E. J. Laschinger, G. D. R. MacCarthy, H. Meyer, S. Morison, P. T. Morrisby, W. H. Odgers, F. B. Ogle, J. F. Pyles, E. T. Rand, L. J. Robinson, W. H. Roe, A. Salkinson, H. A. Scarf, W. Sharp, S. Shlom, T. W. Simmons, R. S. Stokes, W. H. Stout, C. T. Sydenham, J. A. Taylor. A. Thomas, K. Tonnesen, O. Tonnesen, J. F. Walker, F. W. Watson, and L. J. Wilmoth.

12 Associates and Students: Messrs. J. Cronin, C. L. Dewar, G. H. Little, N. Newland, H. B. Powter, A. Short, H. Stadler, C B. Surmon, A. M. Thomas, W. E. Thorpe, J. M. Thorburn, and C. Toombs.

15 Visitors, including two ladies, and Fred. Rowland. Secretary.

The minutes of the previous meeting, as printed in the March *Journal*, were confirmed.

The President: I am asked to apologise for the absence of Dr. Moir, who is unfortunately ill in bed, otherwise he would have been here.

NEW MEMBERS.

Messrs. W. H. Stout and J. A. Taylor were appointed scrutineers, and after their scrutiny of the ballot papers, the President announced that all the candidates for membership had been unanimously elected, as follows:—

ADAMS, ERNEST LAWRENCE, Village Deep, Ltd., P. O. Box 1145, Johannesburg. Mining Engineer.

BENTLEY, FREDERICK WILLIAM, Robinson Deep G. M. Co , Ltd. P. O. Box 4648, Johannesburg. Electrical Engineer.

BOWNESS, ARTHUR JAMES, Ferreira Deep, Ltd. P. O. Box 5977, Johannesburg. Cyanide Foreman.

CARIS, SYDNEY, Ferreira Deep, Limited, P. O. Box 5977, Johannesburg. Cyanider.

FERRIS, GUY GRINDLEY, Ferreira Deep, Ltd., P. O. Box 1056, Johannesburg. Smelter.

GRAMMELL. PAUL, Pajakombo *via* Pedang, West Sumatra, Dutch East Indies. Mining Engineer.

LOWDON, GEORGE WEBSTER, Moodie's G. M. and Exploration Co., Ltd. P.O. Box 117, Barberton. Joint Manager.

MAC GREGOR, ROBERT DUNCAN, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Electrical Engineer.

MACLATCHY, ALLAN GILMOUR, South Rand Exploration Co., Ltd., Balfour. Mine Manager.

PRYCE, LESLIE, Simmer & Jack Proprietary Mines, Ltd., P. O. Box 192, Germiston. Resident Engineer.

ROBERTSON, DAVID, Treasury G. M. Ltd., Cleveland. Battery Manager.

SCOTT, LAWRENCE, East Rand Proprietary Mines, Ltd., P. O. Box 56, East Rand. Mechanical Engineer.

SMITH, JOHN ARTHUR, Ferreira Deep, Ltd., P. O. Box 5977, Johannesburg. Cyanider.

WATSON, JAMES HENRY ARNAND, Robinson Deep G. M. Co., Ltd., P. O. Box 1488, Johannesburg. Electrician.

WEBSTER, DAVID, Robinson Deep G. M. Co., Ltd., P. O. Box 6316, Johannesburg. Electrician.

The Secretary: Since the last meeting of the Society the following associate has been admitted:—

BELL, GUY FALKARD, Ferreira Deep, Ltd., P. O. Box 5977, Johannesburg. Cyanider.

GENERAL BUSINESS.

Prof. G. H. Stanley (*Member of Council*): I wish to exhibit a little silica parting apparatus which Mr. Wm. Cullen has kindly presented for use in the assay laboratory of the College. You will remember that some time ago he brought up some specimens of silica ware, which were made by a newer process, and which were not quite so satisfactory. This specimen is made by the oxy-hydrogen process, with the result that it is as clear as glass. It is a beautiful piece of workmanship, and as it will stand a temperature above the melting point of platinum, it will be appreciated by our members.

A Member: What is the price of it?

Prof. G. H. Stanley: I am sorry I have no idea of the price, Mr. Cullen having only sent it in to-day.

The President: Our thanks are due to Prof. Stanley for showing us this interesting piece of apparatus. Undoubtedly, if it is as efficient as platinum, it will prove to be very much cheaper.

BATTERY AND CYANIDE GOLD SMELTING.

The President: I have a little information to bring forward which is rather belated. It is with regard to a paper read before the Society by Mr. A. Thomas, on battery and cyanide gold smelting, to the discussion of which he replied in the December *Journal*, p. 192 He asked for various particulars connected with some remarks I made upon his paper. As one of the retorts has just been replaced, I have been able to complete these particulars, and I will give them to you in order that they may be published for the information of Mr. Thomas and others. He asked me to let him know the actual consumption of coal used in retorting 5,000 oz. of amalgam, and various other details, also the size of the flues, chimneys and grate area. This is the information: Amalgam taken 4,984 oz. The time taken for retorting, 5 hours; loss on retorting. ·6%; total loss, ·8%; coal (a good ordinary Middelburg coal) used was 201 lb., costing 2s. 8·4d. The grate area was 30 in. × 23 in., or 690 sq. in., size of chimney 10 in. diameter, height about 25 ft., size of flue openings—two centre ones, 3 in. × 2 in., two end ones, 6 in. × 2 in.

ANNUAL REPORT OF COUNCIL.

The Secretary read the Annual Report and Financial Statement, as follows:—

Your Council has great pleasure in placing before you its report upon the work of the past year, together with the usual Statement of Accounts for the twelve months ending the 1st June, 1909.

The Council is glad to record the fact that the depression which has overclouded South Africa is passing away, and that the Society, in common with other institutions, may be said to be sharing, in a minor degree, in the period of prosperity which this Colony is entering on, as the Budget placed before the Transvaal Parliament the other day by the Colonial Treasurer clearly shows. The same care and attention which was practised by the Finance Committee last year, has been paid to the Society's finances during the past year, and it is with some degree of satisfaction that the Council is able to place before you an even better statement of accounts than that presented at the last annual meeting.

The revenue for the year shows an increase, over the last accounts, of £64 8s. 8d., due chiefly to the increase which was made last year in the Associate's subscriptions, inasmuch as, owing to a number of subscriptions remaining unpaid, the receipts from members show a decrease of about £39. On the other side, the general expenditure has been reduced in nearly every item, and shows a saving of over £23, after making provision for doubtful debts. The *Journal*, in which the Society takes a justifiable pride, shows an increase in cost of about £76, due partly to an increase in size of 40 pages for the year, and partly to the increasing use of illustrations made by authors of papers and notes. The surplus of revenue over expenditure shows an increase over last year of £26 18s. 5d.

Turning to the Balance Sheet, it will be seen that our cash assets exceed our immediate liabilities by about £95, which is satisfactory. The value of the stock in hand of volumes of transactions shows an increase of about £95, due to the addition of Vol. IX. The sale of the back volumes has fallen off considerably during the past year, and it is hoped that those members who have not yet completed their sets of these useful volumes, dealing as they do with the early history of metallurgy on the Rand, will take an early opportunity of doing so. The item sundry debtors is somewhat increased by outstanding accounts for advertisements, which will be mostly liquidated during the coming month.

The accounts show a striking increase compared with the Revenue and Expenditure accounts for 1898. The Revenue during that year was £249 18s., and the Expenditure £116 0s. 9d. Our balance this year would have been more if we had not had the expense of moving into and furnishing new offices.

The amount of outstanding subscriptions which, in accordance with our usual custom, is not included in the statement—a very safe practice—is £301 17s. 6d. Since this amount was arrived at, it is gratifying to note that several members have paid their arrears, and the Council would again urge upon all members the desirability of the prompt payment of their subscriptions, which fall due and are payable on the 1st of July of every year.

TABLE I.

	June, 1895.	June, 1896.	June, 1897.	June, 1898.	June, 1899.	June, 1902.	June, 1903.	June, 1904.	June, 1905.	June, 19. 6.	June, 1907.	June, 1908.	June, 1909.
Hon. members ..	5	5	5	8	14	9	13	18	18	16	17	18	21
Life members ...	—	1	1	1	1	1	1	2	3	4	7	7	8
Members ..	49	81	101	197	281	291	441	639	680	680	683	618	630
Associates ...	—	—	—	180	238	243	281	338	321	314	342	357	342
Students ..	—	—	—	—	—	—	—	4	8	4	7	18	20
	54	87	107	386	534	544	736	1,001	1,030	1,018	1,056	1,018	1,021

TABLE II.

DISTRIBUTION.	MEMBERS.			ASSOCIATES.	STUDENTS.	TOTAL.
	Honorary.	Life.	Ordinary.			
Transvaal (Witwatersrand)	4	1	406	153	19	583
Do. (outside districts)	1	—	38	32	—	71
Rhodesia	—	—	58	40	—	98
Cape Colony, O.R.C., and Natal	1	—	7	2	—	10
British East & West, German S.W. & Portuguese E. Africa	—	—	4	—	—	4
British Isles	11	4	47	22	1	85
Australasia	—	1	9	23	—	33
Dominion of Canada ...	—	—	—	2	—	2
India and Burma	—	—	1	5	—	6
Europe	—	—	4	5	—	9
United States of America ...	4	1	22	27	—	54
Central and South America	—	—	3	1	—	4
Mexico	—	—	8	11	—	19
Borneo	—	—	—	1	—	1
Dutch East Indies ...	—	—	11	7	—	18
China and Korea	—	—	—	2	—	2
Federated Malay States ...	—	1	3	1	—	5
Addresses Wanted ..	—	—	9	8	—	17
TOTAL ...	21	9	630	342	20	1,021

The Council has again to express its regret that it has been obliged to strike off the roll 51 members and 34 associates, in respect of non-payment of subscriptions for two years and over. Every consideration is given to those who notify the Secretary that they are temporarily unable to pay, but in practically all the above cases the numerous notices and requests for payment have been totally ignored, thus putting the Society to useless expense.

During the year 83 Members, 47 Associates and 5 Students were elected and admitted, 9 of the new members having been associates who have transferred, and 2 were transferred from student to associate. The resignations of 16 members, 16 associates and 1 student have been accepted with regret, while the loss by death, which has been deeply regretted, has been as follows :—

Three members : Messrs. J. H. Combellack, J. W. Kilby and J. Laurie Hamilton.

Three associates : Dr. W. I. Moir ; Messrs. A. W. K. Pierce and G. P. W. Rofe.

The tables of membership, as usually presented to you, are given herewith, and show that despite the large number of members struck off, the total membership has increased by three.

Twelve Ordinary General Meetings were held during the past year, at which many important and interesting papers were submitted and discussed, and the ordinary business of the Society transacted, while two Special General Meetings of members were held for the alteration of rules.

The following is a list of the papers submitted during the year :—

July, 1908. — The President's (Mr. R. G. Bevington) Inaugural Address.

"Battery and Cyanide Gold Smelting," by Mr. A. Thomas.

August, 1908.—"Reminiscences of the Early Rand," by Mr. M. H. Coombe.

September, 1908.—"Note on a Problem during Shaft Sinking," by Mr. Chas. B. Saner.
"Zinc Dust Precipitation at Cerro Prieto," by Mr. Robert Linton.
"Small Mines of Rhodesia," by Mr. B. I. Collings.

October, 1908.—"The Patio Process," by Mr. C. Perez Duarte.
"The Theory of Blasting with High Explosives," by Mr. E. M. Weston.

November, 1908.—The Silver Coating of Amalgamating Plates" (Note), by Mr. W. A. Caldecott.
"The Gases Resulting from the Use of High Explosives," by Mr. W. Cullen.
"Description of Ore Treatment at the Giant Mine, Hartley District, Rhodesia," by Mr. R. C. H. Cooke.
"A Few Notes on Cupelling Gold-Lead Bullion," by Mr. Geo. Melvill.

December, 1908.—"Gold Crystals" (Note), by Mr. A. McA. Johnston.
"Testing of Safety Fuse by X Rays" (Note), by Mr. Jas. Thomas.
"Routine Assaying on a Westralian Mine," by Mr. W. B. Blyth.
"Grading Assays and Grinding Efficiencies," by Mr. Arthur Yates.

January, 1909.—"Precipitation from Cyanide Solutions by Zinc Shavings and Dust: A Comparison of Results and Costs," by Mr. Allen J. Clark.
"White Labour in Mining," by Mr. Tom Johnson.

February, 1909.—"A Cheap Platinum Parting Apparatus" (Note), by Prof. G. H. Stanley.
"The Estimation of Carbon Monoxide in Mine Gas," by Dr. E. H. Weiskopf.
"The Distribution of the Gold Produced on the Rand," by Dr. T. Kirke-Rose.

March, 1909.—"Improved Glasses for Testing for Potassium" (Note), by Dr. James Moir.
"The Pilgrims Rest Gold Fields and Mining Methods," by Mr. J. Moyle-Phillips.
"Notes on Precipitation," by Mr. Mather Smith.

April, 1909.—"Covers for Extractor Boxes" (Note), by Mr. K. L. Graham.
"A Low-Fall Sands-Return Contrivance," and "Counter-Weighted Blind for Sands Collecting Vat" (Notes), by Mr. C. W. Dowsett.
"The Precipitation of Gold from Solution by Carbonaceous Matter" (Note), by Mr. W. A. Caldecott.
"South African Coals and their Economics," by Mr. A. J. Andrews.
"Suggestions for a New Atomic Theory," by Dr. James Moir.

May, 1909.—"Stope Measurements," by Mr. Otto S. Tonnesen.
"The Influence of Moist Air on Quicklime," by Mr. Jas. Gray.
"Researches upon the Telluride Gold Ores of Cripple Creek, Colorado," by The Portland Metallurgical Society (edited by Mr. Thos. B. Crowe).

June, 1909.—"The Effect on Amalgamation of Different Intervals of Time between the Dressings of Plates," by Mr. G. O. Smart.
"The Smelting of Gold Precipitates and Bullion with Oil Fuel," by Mr. Arthur Yates.

The average attendance at the meetings has been 68, which is slightly over that of the previous year.

Three excursions have taken place during the period under review, and were greatly appreciated, viz., a visit to the Mdderfontein Dynamite Factory, to the Premier Diamond Mine, and to the surface works of the Simmer Deep, Ltd., all of which were of great interest and very instructive to those who participated.

The Annual Dinner was again the usual success, and the speech made by Sir Percy FitzPatrick, one of our Hon. Vice-Presidents and a delegate to the Union Convention, was of great political interest.

Twelve Council Meetings were held, at which the average attendance was 12. The following is a list of the attendances:—

Mr. R. G. Bevington	12	Mr. K. L. Graham	7
Dr. J. Moir ...	5	,, C. B. Kingston	4
Mr. A. McA. Johnston	12	,, A. Richardson...	8
,, S. H. Pearce ...	1	,, C. B. Saner ...	8
,, W. A. Caldecott	6	,, G. O. Smart ...	11
,, W. Cullen ..	1	Prof. G. H. Stanley	11
,, E. H. Johnson	1	Mr. J. E. Thomas...	9
Prof. J. Yates ...	1	,, A. Whitby ...	8
Mr. J. Littlejohn ...	1	,, H. A. White ...	12
,, E. H. Croghan	8	Prof. J. A. Wilkinson	12
,, W. R. Dowling	9		

Regular monthly meetings have been held of the Finance and Publications Committees, whilst the Editorial Committee, whose work is very onerous and mostly accomplished apart from meetings, has met when required. The work of these Committees, which has taken up a great deal of the time of the members serving upon them, deserves the grateful thanks of the Society. The members of all the Committees have been indefatigable in their labours, and it should be remembered that many of them have had to come considerable distances at some personal inconvenience, to attend these meetings.

The new building of the Transvaal University College was completed about the end of last year, and satisfactory arrangements having been made, your Society, together with others, removed its headquarters to a commodious office (which has been well furnished at a cost of £86) on the ground floor of the College Building. After a period of some fifteen years, during which time the monthly meetings of the Society were held in the comfortable Council Chamber of the Chamber of Mines, it was decided to hold the meetings in the Lecture Theatre of the College, which, if not quite so comfortable, nor so good in its acoustic properties as the old meeting place, is better suited and is moreover adequately equipped for the purposes of practically demonstrating and illustrating technical and scientific papers.

On the same floor as the office is also situated the Seymour Memorial Library, which is being utilised to a much greater extent than formerly, and the Technical Societies' Common Room, which has been set aside by the College Council as a meeting place, writing room, etc., for the members of the several societies. Up to the present the Common Room has been very little used, and it is greatly to be hoped that the members will in future make more use of it.

The question of Closer Co-operation between the several technical societies was discussed at the last Annual Meeting, but in the absence of any definite proposals, the Council has nothing fresh to lay before you at this meeting. Prof. Orr, in his valedictory address to the Institute of Mechanical Engineers last week, referred at length to the question of Federation, and it is understood that the Council of that Institute will move in the matter shortly. The Society may be assured, however, that the incoming Council will jealously guard the interests of our Society in whatever steps may be taken, and, of course, any proposals for Federation will be laid before you for decision previous to committing the Society to any particular course of action in this direction.

During the year the Mines Trials Committee, which has for its object the practical testing of inventions and improvements in matters of metallurgical practice, extended its scope so as to include inventions in mining practice and mechanical appliances. This extension of the work of the Committee, will, the Council feels, be of infinite advantage to the small inventor and the mining industry generally.

At the beginning of this year the Government introduced new mining regulations providing for the greater safeguarding of the gold during the process of extraction until handed to the banks. By the courtesy of the Government Mining Engineer, an advance copy of these regulations was considered by a Committee of your Council, and although some of the regulations were considered to be apparently irksome to carry out in practice, and possibly of little practical value, it was considered advisable not to press for any amendment in them until the effect of the regulations, under actual working conditions, had been thoroughly tested.

The Government has recently nominated a Commission to go into the question of the effect upon the country in general and the industry in particular of the proposed new electrical power-schemes now before Parliament. Your Council has been requested to lay evidence before the Commission, and the matter is now receiving the attention of a sub-committee of the Council, with a view to a statement being prepared to be submitted to the Commission.

The joint standardisation committee on cement testing and other kindred matters, referred to in the last report, upon which your Society has representatives, is still considering the question, but nothing has yet been done which can be reported to you.

Your Secretary has been the Society's representative upon the Rand Regiments Monument Committee, which is now taking active steps towards the erection of a monument in Johannesburg to the memory of all Rand men who fell on the British side during the war. That Committee is now inviting subscriptions for this object, and it is hoped that before long a suitable memorial will be erected. The Society, as you know, has its own memorial which is in the Chamber of Mines, but the hope is expressed that members, who have not already contributed to the Rand Regiments Monument, will send any donation they may deem desirable to the Secretary, who has a list open in the office for the purpose.

Several amendments to the Constitution and Rules have been made during the year. In view of certain correspondence between members of Council and others in distant parts, it was considered that it might be to the advantage of the Society if facilities were afforded for the formation of sections of the Society on similar lines to those successfully adopted by the Society of Chemical Industry. Rules giving effect to the suggestions made by Mr. W. A. Caldecott on this subject, were adopted at the December meeting, but up to the present the provisions of these Rules have not been complied with by any members, although there has been considerable correspondence on the subject with members in Rhodesia and Mexico.

The Rules regarding the admission of student members have been amended in order to provide for the large number of learners and apprentices at present working on the mines, who are not in a position to enable them to pay anything but the nominal student fee of half a guinea. Steps are being taken to advise all the mines of this extension of the privileges of the Society, and the Council trusts, that before the next Annual Meeting, a large number of these prospective members will have taken the first step by joining as students.

As referred to in the last Annual Report, an extension of the Society's work in other parts of the world was taken in hand by the election, by the Council, of Corresponding Members of Council. Owing to the distances and the time taken in correspondence, it is not yet possible to accurately gauge the effect of this scheme, but judging by the sympathy and efforts of those gentlemen who have kindly undertaken to act for the Society in distant lands, the success of the scheme is assured. Since the foregoing was written, news has been received that both New Zealand and India will be added to the list on the front page of the *Journal*, and our thanks are due to all of those gentlemen for the work they are doing on our behalf. In this connection it is gratifying to make formal acknowledgement in this Report of the action of the Institution of Mining and Metallurgy in electing the President of your Society for the time being, as one of its Corresponding Members of Council, an honour which is highly appreciated.

To revert to one or two matters of local interest, the Council, for the past eight months, has given effect to the suggestion to distribute at the monthly meetings, proof copies of papers to be read. This action appears to have met with the approval of members, and as the cost is very moderate the practice will be continued.

A suggestion was made to your Council to reduce the size of the *Journal* pages, and so to arrange the matter that each paper could be extracted and filed for reference under subjects. After careful consideration of the idea, which has undoubted advantages, the Council felt that the alteration would not appeal to the majority of its members. It was also decided at the beginning of the year, that, in view of the increased Associate subscription (which has been accepted cheerfully by those affected) to increase the price of single copies of the *Journal* to non-members from 2s. to 3s.

In conclusion, your Council desires to place on record its thanks to the Chamber of Mines, and to the Council of the University College, for the use of rooms for the monthly meetings; to Messrs. van Hulsteyn, Feltham and Fry, the Hon. Solicitors, for legal advice and assistance; to your Auditor, Mr. Alex. Aiken, for the really arduous work entailed (performed quite honorarily) in the auditing of your accounts; and to your Hon. Treasurer, Mr. James Littlejohn, and to Mr. A. Whitby who acted in his stead during Mr. Littlejohn's serious illness. Your Council also desires to record its regret that Mr. T. Lane Carter, one of your Vice-Presidents, and Mr. C. B. Kingston, another member of your Council, will not be able to stand for election this year owing to their absence from the country; and to express the acknowledgements of the Society to the local Press for the space accorded and the publicity given to the proceedings of the Society and notices of its activities.

The President: You have heard the Report and Balance Sheet for the year. I beg formally to move that the same be adopted.

Mr. E. J. Laschinger (*Member*) seconded the motion.

Mr. H. A. White (*Member of Council*): I want to call attention to one little point in the Report regarding the list of attendances of Members of Council. I notice that all our Past-Presidents, except one, were content with a single attendance.

Mr. J. Gray (*Member*): There is one item I notice has been omitted, namely, thanks to Mr. Rowland, our Secretary. He is too bashful himself to include it in the Report, but I think it only needs mentioning, for him to receive a very hearty vote of thanks.

The vote of thanks was carried by acclamation.

THE WAR MEMORIAL.

Mr. E. H. Johnson (*Past-President*): There is an item mentioned in the Report with reference to the memorial to those who fell during the war. Since we have moved to our new home, do you not think our little memorial might be moved out of the shockingly bad light it is in at present, at the Chamber of Mines? I think we should obtain the consent of the Council of the University to have it placed in the entrance hall of the College.

The President: The matter has been under the consideration of the Council of the Society, and it was thought it might be advisable to leave it where it is for the present, as the Government and Municipality are considering the matter of erecting a number of large buildings, Town Hall, Museum, and so forth. When these buildings are completed, a final and permanent home may be found in one of them, perhaps in the new Town Hall

The Report and Financial Statement were adopted.

THE CHEMICAL, METALLURGICAL AND MINING SOCIETY OF SOUTH AFRICA.

(INCORPORATED MAY, 1907.)

REVENUE ACCOUNT for year ending 1st June, 1909.

Dr.

1907-8. £ s. d.	EXPENDITURE.	1908-9. £ s. d.	£ s. d.
	To CHARGES (including Postages, Attendance, Bank Charges, etc.) ...	187 18 0	
(202 9 4			
159 15 11	,, PRINTING AND STATIONERY ...	157 2 0	
90 12 0	,, RENT AND LIGHTING	67 18 0	
386 0 0	,, SALARIES...	386 0 0	
11 2 0)	,, DEPRECIATION ON FURNITURE ...	7 8 0	
	,, PROVISION FOR DOUBTFUL DEBTS	20 0 0	
			826 6 0
849 19 3			
15 5 7	,, *Society's Medals.*		
	,, SOCIETY'S JOURNAL		
(918 16 11	Cost of Reporting, Printing and Sundries	995 12 5	
408 11 10)	*less* Received for Subscriptions and Advertisements ..	409 8 3	
509 15 1			586 4 2
233 4 5	,, BALANCE, surplus on year's working		260 2 10
£1,608 4 4			£1,672 13 0

Cr.

1907-8. £ s. d.	REVENUE.	1908-9. £ s. d.	£ s. d.
	By SUBSCRIPTIONS—		
(1,174 6 0	Members	1,135 11 6	
353 6 6	Associates	455 19 3	
6 16 6)	Students	8 13 3	
			1,600 4 0
1,534 9 0			
59 17 0	,, ENTRANCE FEES		72 9 0
13 18 4	,, *Surplus Receipts over Expenditure—Annual Dinner and Nigel Visit.*		
£1,608 4 4			£1,672 13 0

BALANCE SHEET, as at 1st June, 1909.

1907-8. £ s. d.	LIABILITIES.	1908-9. £ s. d.	£ s. d.
(3 16 2	To SUNDRY CREDITORS	136 11 6	
53 11 0	,, SUBSCRIPTIONS paid in advance ...	27 16 6	
27 0 0)	,, PRIZE FUND	6 0 0	
			170 8 0
84 7 2			
	,, REVENUE ACCOUNT—		
(998 2 10	Balance at June 1, 1908 ...	1,231 7 3	
233 4 5)	Add surplus as per Revenue Account, June 1, 1909 ...	260 2 10	
			1,491 10 1
1,231 7 3			
£1,315 14 5			£1,661 18 1

1907-8. £ s. d.	ASSETS.	1908-9. £ s. d.	£ s. d.
(203 15 6	By CASH AT BANK	174 15 2	
4 11 0)	,, CASH IN HAND	91 0 3	
			265 15 5
208 6 6			
15 0 0	,, OFFICE FURNITURE		86 0 0
(99 19 0	,, TRANSACTIONS, VOL. II.—Stock in hand	89 7 9	
32 8 8	,, ,, VOL. III. ,,	25 10 2	
226 10 0	,, ,, VOL. IV. ,,	199 16 3	
119 3 5	,, JOURNAL, Vol. V., Stock in hand ...	115 1 11	
138 11 9	,, ,, Vol. VI. ,, ...	134 11 3	
145 15 0)	,, ,, Vol. VII. ,, ...	146 6 0	
82 0 0	,, ,, Vol. VIII. ,, ...	136 2 1	
	,, Vol. IX. ,, ...	93 0 0	
844 7 10			939 15 5
(263 16 7	,, SUNDRY DEBTORS	372 10 0	
15 16 6)	*less* Provision for Doubtful Debts ...	2 2 9	
248 0 1			370 7 3
£1,315 14 5			£1,661 18 1

I have examined the foregoing Statements of Account, compared them with the Books and Vouchers of the Society, and I find the statements to be true and correct, as shown thereby.

Sgd. ALEX. AIKEN, *Incorporated Accountant,* } *Hon. Auditor.*

JOHANNESBURG, 19th JUNE, 1909.

(Signed) J. LITTLEJOHN, *Hon. Treasurer.*

(Signed) FRED. ROWLAND, *Secretary,*
Fellow Chartered Inst. Secretaries.

MEXICAN INSTITUTE OF MINING AND METALLURGY.

The President: I should like to mention that we have received notice of the formation in Mexico of a Mexican Institute of Mining and Metallurgy, and I think it is only right and proper, considering this new institute is going to work on lines similar to our own, that we should extend to them our hearty greetings and best wishes for the success of the institution.

It is not my intention to-night to inflict upon you a valedictory address. I shall follow the excellent example set by my predecessor, and leave the field open for the incoming President to select the matter for his inaugural address. Although our membership is very large, there are still large numbers of workers on the reef who certainly ought to belong to the Society, and I regret to say that there are very many, who although they are occupying very good positions and quite able to pay the subscriptions to this Society, which after all amounts only for a member, to about 1½d. a day, and for an associate member 1d. a day, yet these men content themselves with a cheap form of acquiring knowledge and information, by borrowing our *Journals* from other members, and reading them. If these people find that our *Journals* contain matter which is interesting to them, and from which they can extract information and instruction, I certainly think they ought to support our Society and strenghten it. For after all, it must not be forgotten that it is greatly due to the labours of the members of the Society, from the highest to the lowest, that this magnificient mining industry has been brought to such a stage as it now is in, and by this means, finding employment for such a large number of those whose work comes under one or other of the branches dealt with by this Society.

In taking leave of you to-night, as your President, I need hardly tell you I do so with much regret. My year of office has been to me a most pleasant one, and I wish to take this opportunity of thanking you all for the loyal and cordial support you have extended to me. I should also like specially to thank the members of the late Council, many of whom I hope to see re-elected. I am sure that my successor, whoever he may be, will appreciate your action, and I am sure he will find that the meetings of the Council in the future will be characterised by that zeal in the interests of the Society, and by the same friendship which has been such a feature of the past year. I cannot close my remarks without referring to the work which has been done by our Secretary, Mr. Fred. Rowland. His labours have been very arduous, and especially so with regard to producing the *Journal*, which, as you have seen in the Report, has been a feature of the Council's work. I am sure it is a *Journal* of which we must all be proud, for I am sure there is nothing approaching it in South Africa. I wish to thank Mr. Rowland, personally, for the arduous work he has done, and for the great assistance he has given me in other matters. I wish also to say that his work has been very much esteemed by your Council, who at their last meeting had the pleasure of voting him a bonus of £50 for his excellent work. With these few remarks I will take my leave of you as President.

ELECTION OF OFFICERS AND COUNCIL.

The President: A matter which you are all eagerly waiting for is the declaration of the election of officers for the coming year. The following is the report and the result of the voting:—

Johannesburg, 19th June, 1909.

THE CHAIRMAN,
ANNUAL GENERAL MEETING OF MEMBERS.

Dear Sir,—We have to report having received 194 ballot papers, of which 5 were invalid in respect of voting for Vice-Presidents and Council.

The result of the voting is as follows:—

PRESIDENT:

Mr. A. McA. Johnston.

VICE-PRESIDENTS:

Dr. J. Moir.
Mr. W. R. Dowling
Mr. C. B. Saner.

HON. TREASURER:

Mr. James Littlejohn (*unanimously*).

MEMBERS OF COUNCIL:

1 Prof J. A. Wilkinson
2 Mr. K. L. Graham.
3 ,, A. Richardson.
4 ,, G. O. Smart.
5 Prof. G H. Stanley.
6 Mr. F. F. Alexander.
7 Mr. H. A. White.
8 ,, A. Whitby.
9 ,, E. J. Laschinger
10 ,, E. H. Croghan.
11 ,, J. E. Thomas.
12 ,, M. Cullen.

Yours faithfully,
Sgd. JAS. GRAY,
,, RALPH STOKES,
,, L. J. WILMOTH,
,, A. DONALD VINEY,
,, S. NEWTON,
Scrutineers.

The President: I must heartily congratulate you upon your choice. In Mr. McArthur Johnston you have a man of high scientific attainments, and of sterling character. I am sure he will do great things for the Society. I am very glad that you have re-elected so many of the late Council, as they have been very hard workers.

Mr. A. McArthur Johnston (*Vice-President*): Mr. President, ladies and gentlemen, it affords me great pleasure in having to thank you for the honour you have done me. Unfortunately I am not prepared to make a speech to-night, but I hope at the next meeting to show my appreciation more fully. I congratulate you on the good sense you have shown in re electing so many of our late members of the Council. They have done the work well in the past, and I am sure they will assist me in the work we have to tackle in the future.

Mr. C. B. Saner (*Member of Council*): In the absence of Dr. Moir, and on behalf of Mr. Dowling and myself, I wish to thank you very heartily for electing us as Vice-Presidents. I also wish to move a vote of thanks to the scrutineers. It took them a long while to get through their arduous and responsible work, and I think it is only due to them for the excellent manner in which they have carried out their duties.

The vote of thanks was carried unanimously.

Mr. J. Gray (*Member*): I feel that the duty of returning thanks on behalf of the scrutineers falls to me, as this was the fifth occasion on which I have acted as scrutineer. I have also on behalf of the scrutineers to thank Mr. Rowland, who on behalf of the Society, so generously sustained us during our arduous labours.

VISIT TO SIMMER DEEP, LTD.

The President: Since our last meeting, as you have seen in the *Journal*, the Society had the pleasure of visiting the surface works of the Simmer Deep, Ltd., an excursion to which I am sure we all looked forward for a very long time, and I should like to formally place on record the appreciation of the members of that visit and of what we saw and learned, and also to record our hearty thanks to the Consolidated Goldfields of South Africa, and to the management and staff of the Simmer Deep, for so kindly allowing us to visit the property and for showing and explaining everything to us, and for entertaining us in the way they did.

The vote of thanks was carried by acclamation.

Mr. E. J. Laschinger (*Member*): Votes of thanks are in order, but I am certain the vote I would like to propose just now is not one which is simply a matter of routine, namely, a vote of thanks to the retiring President and Council and to the officials for the very excellent work they have done during the year. It is reflected in the statements of accounts and the balance sheet. Every Society looks to its Council for results, and the results have been most excellent. Notwithstanding the taking away of the membership of those who did not pay their subscriptions, they have nevertheless added to the list of members, and carried us through the past year—a year of great depression when everyone has been rather hard pushed for money—very successfully and given the Society a very fine statement of accounts. I think the Society has never been in a more flourishing condition than it is at the present moment. I will ask the members to pass this vote of thanks by acclamation.

The President: Speaking on behalf of myself and the late Council, I thank you very heartily indeed for the appreciation which you have shown of our work. I am sure it has been a pleasure to both myself and the Council, and what the Council has done in the past I am sure they will continue to do in the future, that your new President will receive their fullest support, and that he will at the end of his term have the same feelings towards the Society and towards the members of the Council as I have.

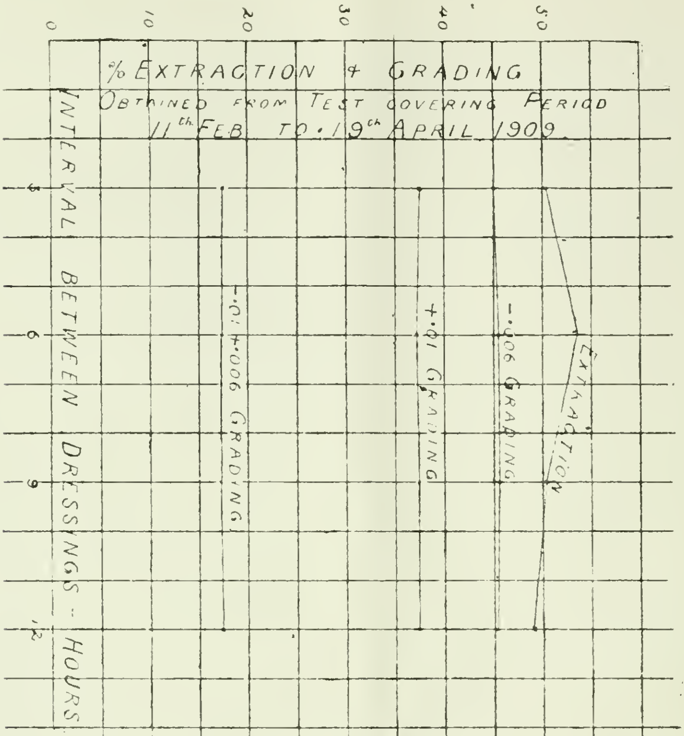
It has been thought by the Council that it would perhaps be a pleasant innovation in our proceedings if at 9 o'clock we had an interval of five or ten minutes when coffee could be served. It is quite an innovation, of course, it is a matter of but small expense to the Society, and I would like members to express their opinion about it The actual expense to the Society for each meeting, taking the average attendance, would be about a sovereign. I would like someone to bring forward a motion on the subject a little later on.

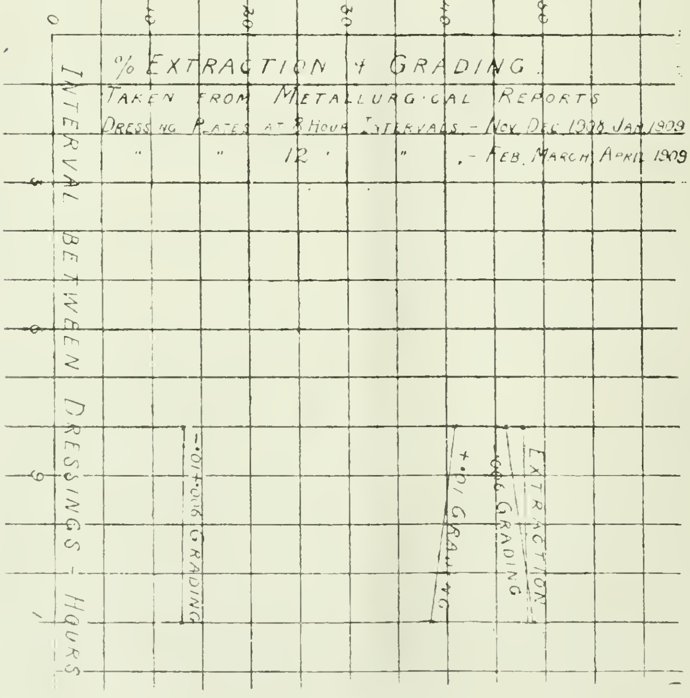
After coffee had been served, which afforded an interval of informal conversation, the following paper was read :—

THE EFFECT ON AMALGAMATION, OF DIFFERENT INTERVALS OF TIME BETWEEN THE DRESSINGS OF PLATES.

By G. O. Smart (Member of Council).

Following on the successful installation of tube mills in reduction works practice and the consequent coarse crushing of ore by the stamps, it appeared to me that longer intervals of time between the dressings of battery plates would not materially affect the percentage of gold recovered by amalgamation, as owing to the coarse sand and increased tonnage passing over the plates, the surface remains bright, and a comparatively small amount of black sand is caught thereon. Even should the extraction be slightly less, it might still be advantageous to dress plates at longer intervals, seeing that there is a secondary amalgamation to follow on the tube mill plates.


% Extraction & Grading
Obtained from Test covering Period
11th Feb. to 19th April 1909.
0
10
20
30
40
50
3
6
9
12
Interval between Dressings – Hours
-·01 +·006 Grading
+·01 Grading
-·006 Grading
Extraction


% Extraction & Grading.
Taken from Metallurgical Reports
Dressing Plates at 8 Hour Intervals. – Nov. Dec. 1908. Jan. 1909
" " 12 " " – Feb. March April 1909
0
10
20
30
40
50
3
6
9
Interval between Dressings – Hours
-·01 +·006 Grading
+·01 Grading
-·006 Grading
Extraction

Under the New Gold Law Regulations, covers with double locks have to be used on all mill plates. This necessitates the presence of two white men each time these covers are removed, and as every dressing entails extra labour and the withdrawal of shiftsmen's attention from the efficient running of the mill, it is clearly evident that any reduction in the number of plate dressings required would be beneficial to general mill practice, provided the total extraction be not detrimentally affected.

Trials were accordingly undertaken to prove the truth of this contention, or otherwise, with the following results:—

During November, December, 1908, and January, 1909, all plates in the mill were dressed at eight hour intervals, and in February, March and April, 1909, at twelve hour intervals.

Table 1. gives a comparison of the results obtained for the two periods involved.

TABLE I.—EXTRACTION ON MILL PLATES.

Date.	Heads dwt.	Tails dwt.	Extraction dwt.	Extraction %	Grading.		
					+ ·01.	-01 + ·006	- ·006.
Nov., Dec., 1908, Jan., 1909, eight hours between dressings	7·584	3·952	3·632	47·890	40 90	13·03	46·07
Feb., March, April, 1909, twelve hours between dressings	7·359	3·818	3·541	48·118	38·27	12·97	48·77

It will be noticed that the extraction is higher for the period of twelve hour dressings, but on the other hand, and in part explanation of this, the gradings show a finer product passing over the plates during that trial.

A further experiment, covering a period from February 11th to April 19th, 1909, was undertaken to determine the extraction at different intervals of time between the dressings of the plates, and the results are as embodied in Table II. (see next page). From this table it will be seen that the extraction progressively improves up to six hours from the time of dressing plates, and then falls off again, until at the end of twelve hours it has dropped to 49·10%, or ·09% lower than that obtained three hours after dressing. The highest extraction of 53·57%, obtained six hours after dressing, has an original value of 8 38 dwt., whilst for the lowest extraction, viz., 49·10%, obtained twelve hours after dressing, it is only 7·79 dwt., this being the lowest original value in the tables. In addition, it is to be noted that the percentage of + ·01 grade is lowest for the highest extraction.

The results of both tests can best be seen from the diagrams supplied*. As a result of these tests, I have adopted twelve hour intervals between the dressings of mill plates, which appears to be sufficient under the conditions obtaining at the Simmer and Jack, but I would not advocate others doing the same, without making preliminary tests, as the same results may not be obtained in every case.

In conclusion, I may say that the plates must be in good order, and a sufficient setting of amalgam must be left after clean-up. If plates are scraped down to bare copper, a practice only too common in some mills on the Rand, good amalgamation will not be obtained even with dressings every hour.

I am making similar tests on the tube mill plates where there is considerably more black sand. Probably these plates will not run as long as the mill plates without dressing, but from the figures already obtained it would appear that they do not require dressing every four hours. I will be able to give these figures in my reply.

The President: I beg to move a hearty vote of thanks to Mr. Smart for his interesting and instructive paper. I may say that for some few months past I have been dressing plates on the Village Main Reef Mine every eight hours, and I, certainly, have not found any apparent diminution in the yield of amalgam, that is to say, extraction. I have not attempted to go quite so far as the author in dressing every 12 hours, because I wanted to carry the eight hour dressings a little further. It is a very curious fact he mentions with regard to the best extraction obtaining six hours after dressing, and it struck me in thinking over the matter whether this was possibly due to the fact that a freshly dressed plate would be in a very bright and slippery condition, and perhaps might not catch up the small particles of "black sand" which is really a misnomer, as they are really small pieces of pyrites bearing fragments of native gold which just enables them to get caught on the plate. It struck me that a freshly dressed plate might allow these particles to pass over it more easily than after a certain period of time, during which there would have been taking place a gradual accumulation of

---

* See "Economics of Amalgamation," by Justin H. Haynes. —*Mines and Minerals*, Feb., 1909; abstracted in this *Journal*, p. 438—9.

TABLE II.—Plates dressed each day at 6 a.m., and not dressed again during the time of taking samples.

Date.		9 a.m. (3 hours after dressing).				12 noon (6 hours after dressing).				3 p.m. (9 hours after dressing).				6 p.m. (12 hours after dressing).			
		Heads, dwt.	Tails, dwt.	Extraction, dwt.	Extraction, %	Heads, dwt.	Tails, dwt.	Extraction, dwt.	Extraction, %	Heads, dwt.	Tails, dwt.	Extraction, dwt.	Extraction, %	Heads, dwt.	Tails, dwt.	Extraction, dwt.	Extraction,
February 11	...	7·45	4·55	2·90	38·93	6·30	3·80	2·50	39 68	5·25	3·05	2·20	41·90	6·80	3·90	2·90	42·65
12	...	8 05	3·80	4·25	52·80	14·82	6·75	8·07	54·45	14·00	5·75	8 25	58·93	10·70	4·50	6·20	57·94
14	...	9·53	4·60	4·93	51·73	9 70	5·13	4·27	44·02	7·65	4·95	2·70	35·29	10·10	4·35	5·75	56·93
March 18	...	10·57	4·63	5·94	56·20	10·45	5·15	5·30	50 72	8·65	4 57	4·08	47·17	10·50	5·44	5·06	48·19
21	...	10·03	5·25	4·68	46·66	6·25	2 80	3·45	55·20	8·15	3·00	5·15	63·49	6·20	3·37	2 83	45·65
31	...	7·15	3·30	3 85	53·85	5·67	2·32	3 35	59·08	5·10	3·15	1·95	38 24	8·20	4·77	3·43	41·83
April 6	...	4 60	2·38	2 22	48·26	6·20	2·60	3·60	58·06	8·56	3 24	5·32	62·15	5·28	2·28	3 00	56 82
9	...	8·09	3·72	4·37	54·02	7·84	2·92	4·92	62·76	8·28	3·22	5 06	61·11	7·40	3·60	3 80	51·35
16	...	6·68	3 00	3·68	55·09	6·70	2·72	3·98	59·40	6·20	4·80	1·40	22·58	5·84	4·08	1·76	30·14
19	...	7·76	3·84	3·92	50·52	9·92	4 44	5·48	55·24	8·24	4·00	4·24	51·46	6·88	3·36	3·52	51 16
Averages	...	7·99	3 92	4 07	50·48	8·38	3·89	4·49	53·57	8·01	3 97	4·01	50·39	7·79	3·97	3·82	49·10
Grading				Average				Average				Average				Average	
+ ·01	..	38·57	36·47	37·52		37·93	36·27	37·10		37·90	36 80	37·35		38·00	36·71	37·36	
− ·01 + ·006	...	17 26	17 61	17·44		17·46	17·10	17·28		17·29	16 96	17·13		17·53	17·27	17·40	
- ·006	...	44 17	45·92	45·04		44·61	46 63	45·62		44·81	46·24	45·52		44·47	46·02	45·24	

the so-called "black sand," which would assist in holding up other particles travelling down the plate. Then again, later on the extraction falls. Might that not be due to a considerable amount of very fine "black sand" having collected on the plate and coating it to such an extent as to begin to prevent the particles being able to adhere to the plate. It just occurred to me that might afford some explanation.

Mr. W. R. Dowling (*Member of Council*): In seconding the vote of thanks to Mr. Smart for his valuable paper, I would like to quote some results obtained at the joint mill of the Knights Deep and Simmer and Jack East.

In the first series of tests, using 800 mesh screen with ·025 in. aperture :—

At 4 hours after dressing the recovery was						56·9%
,, 8	,,	,,	,,	,,		57·3

In another series with 400 mesh screen and ·035 in. aperture

At 4 hours after dressing the recovery was						44·2%
,, 8	,,	,,	,,	,,		45·5
,, 10	,,	,,	,,	,,		44·8
,, 12	,,	,,	,,	,,		43·7

Immediately after the 12 hour sample was taken the plates were dressed, and a few minutes after re-starting sampled again, when the extraction was 46·1%.

A third series of samples showed :—

At 4 hours after dressing the recovery was						49·5%
,, 8	,,	,,	,,	,,		48·7
,, 12	,,	,,	,,	,,		43·1
and immediately after dressing						48·9

The conclusions drawn from these and other results were that it was perfectly safe to change from the then practised 4 hours dressing, to 8 hours, but that 12 hours was too long for this particular ore, although according to the second series quoted, 10 hours run was possible. The objection to the 10 hour interval was that it did not divide up the 24 hour day conveniently. It was, therefore, decided to dress at 8 hour intervals. Both mills have now been running on these lines for the past six months without any loss either in the mill or the cyanide works.

It is interesting to note the immediate recovery of efficiency of the plates after dressing, as shown in the second and third series of results. These results do not confirm Mr. Haynes' conclusion that the efficiency improves up to a certain time after dressing and then drops again. I admit, however, that I have seen indications of this rise and hope to satisfy myself on the point later.

The advantages of longer intervals between dressings are obvious. In my own case, without asking the amalgamators to do more work than before or putting on additional men, I was able to comply with the regulations requiring two white men to be present at the uncovering and work on the plates. In passing, I would like to express it as my opinion that the work and trouble connected with the covering of plates and boxes was very much overrated, and that the saving of milling time by longer intervals between the plate dressings far more than offsets the trivial delay of removing plate covers.

Mr. Smart deserves thanks from all those engaged in the mining industry, as I understand that quite a number of mills have already extended the four interval to eight, with corresponding increase in milling capacity and consequent profit.

Mr. F. F. Alexander (*Member*): I would like to ask Mr. Smart what he considers a good setting. This is a very important factor in amalgamation, and might be more clearly set forth, particularly with regard to coarse crushing. The ancient practice on these fields has retained a large amount of amalgam on the plates, but with the crushing now obtaining this is impossible. A good setting of amalgam may be either 500 oz. or 50 oz. Either may be good, and possibly both very bad, unless the plate is properly conditioned. Therefore will the author give us some idea of what he means by a "good setting" with a 200 mesh screen?

Mr. G. O. Smart (*Member of Council*): You can steam the plates or scrape close, so long as you avoid exposing copper. However, if after steaming, the plates are scrubbed with blocks of wood covered with screening, coarse sand and mercury being applied to the plate at the same time, and the resulting soft amalgam removed, some difficulty will be experienced in keeping a good amalgamated surface for some time afterwards. I have no objection to steaming plates in moderation, indeed I rather think it does them good.

Mr. H. A. White (*Member of Council*): I would like to add my testimony to that of Messrs. Smart and Dowling.

Ever since the new regulations came into force, requiring battery plates to be covered over by a screen, we have dressed the plates at the May Consolidated Mine every 8 hours, in place of every 4 hours, as was done formerly. No loss in extraction by amalgamation has been experienced, in fact, practically no difference has been observed.

The figures quoted, especially those by Mr. Dowling, scarcely warrant the deduction that improved results are obtained. The differences are small, and when the normal variations can be so great as shown by the author the only sound conclusion to be gathered from his admirable table of results is that no significant difference can be shown between 4 hour and 8 hour dressings. The thanks of the Society are due to him for publishing these figures, which are of especial value at this time.

Mr. A. McA. Johnston (*Vice-President*): Have you tried 12 hour dressings?

Mr. H. A. White: Twelve hour dressings would be more awkward for us, as 8 hour intervals suit the alternation of shifts. The results shown by the author do not, however, encourage experiments in this direction, and I do not propose to make any.

The President: I quite agree with Mr. White as to these very small differences. They are so very small that one can afford to absolutely ignore them. If there had been a difference of 5, 6, or 10%, we might have seen something in them. The differences are so small as to lead us to believe the results are practically the same.

Mr. Hans Meyer (*Member*): May I ask the author what is the average recovery value of the Simmer and Jack? I ask this because I feel that his figures showing his "heads," unless they are exceptional, are very high all through.

Mr. G. O. Smart: I hardly understand the question, and do not exactly know what information is required. If it is the gold value of the Simmer and Jack ore and the extraction obtained therefrom, reference to the published reports of the company is all that is required. It is not my intention to discuss this matter.

Mr. E. J. Laschinger (*Member*): I would like to say one thing in view of what has been said by Mr. White and the President, with regard to the small differences which are said to be really of no account. I should like to congratulate the author on giving us such exceptional figures, working down to three decimal places. From an engineering and scientific point of view we have to deal with definite data, but figures are too often presented in a rough and crude form. If a metallurgist is making experiments he deserves every credit for going into them thoroughly and working things out accurately.

The President: I had no fault to find with the author's figures; it was the result of his figures I was looking at.

Prof. J. A. Wilkinson (*Member of Council*) next read the following paper:—

THE SMELTING OF GOLD PRECIPITATES AND BULLION WITH OIL FUEL.

By Arthur Yates, M.I.M.M. (Corresponding Member of Council).

At the Redjang Lebong Gold and Silver Mine, Sumatra, we have a 70 stamp battery with cyanide works, and our precipitation in the latter is done in the usual zinc boxes, the extractor house equipment including an acid treatment tank for "shorts" only, a filter press, and a calciner, the latter dealing with both the non-acid treated precipitate and the acid treated "shorts." We find that good washing and good calcination materially affect the amount of matte produced in the smelting of the precipitate from the "shorts."

The smelting of both precipitates is carried out in two tilting furnaces, also called Balbach or Fabre du Faure furnaces; these find an application in the metallurgy of zinc, and I believe were first used in connection with the gold precipitates of the cyanide process in Western Australia.

The inside measurements of a retort are: mouth, 6½ in.; greatest diameter, 13¾ in.; bottom diameter, 9½ in.; length, 30 in.; the mouth is closed during smelting with the stand of a No. 50 salamander crucible. The retort sits in the furnace at an angle of 30° to the horizontal.

The furnaces, as supplied by the maker, were bricked in on top and covered with a casting having a central fuel feed door. This arrangement threatened us with considerable work and expense every time a retort had to be changed, also with bad distribution of the fuel and troublesome stoking, besides interfering with the partial turning of the retort, which is necessary every day of service, to prevent cutting through on the metal line. So we improved it by closing in the top with three loose arches of firebrick, the bricks of each arch being clamped together with a strap of 3 in. × ½ in. flat iron with a ring bolt over the centre by which it may be lifted. We also placed a large loose fireclay tile over the neck of the retort within the front plate casting. These adjustments enable the retorts to be changed quickly and without expense. In case a retort should crack or cut through, there is a 2 in. hole, plugged with ore slimes, in the centre of the furnace bottom, through which the metal and slag can be run.

The furnaces are compact, handy and clean in operation; the retort, which projects through the front casting of the furnace, is charged from the outside, and there is none of the heat and trouble which is associated with the reverberatory and pot furnaces. The contents are always under control, and the removal of the charge, by turning a hand wheel which tilts the furnace, is simple and safe. Until April, 1908, the furnaces had been worked with a mixture of coke and charcoal, charcoal alone not giving sufficient heat to melt the charge; the difficulties attending transport here made this fuel expensive, as coke costs up to £15 6s. per ton.

Four "Billow" atomisers, class "FM," made by the National Supply Co., of Chicago, and costing $20·00 each, were ordered and tried in April with excellent results, two being used with each furnace. The furnaces were filled in with firebricks until a space of only 2 in. was left around the sides and bottom of the retort. Sitting on the neck of the retort in front, and supported on a brick in the flue at the back, half of an old retort, cut longitudinally, was fixed as a hood. Just below the retort, and centrally placed in the sides of the furnace two pieces of 3 in. pipe were built into the brickwork and in these the atomisers were placed, their brass shells being protected by a cover of 2 in. piping screwed over them and projecting 1 inch in front.

Kerosene oil is used, and each furnace has a twenty gallon oil storage tank placed 18 ft. above the atomisers, with which they are connected by ¾ in. piping and flexible metallic (steel) tubing. Air at about 40 lb. pressure is delivered through a 2 in. main with ¾ in. branches fitted with ½ in. brass cocks and connected to the atomisers by ¾ in. flexible metallic tubing. To economise heat, and assist the vaporising of the oil, the air supply main enters the stack at the junction of the two furnace flues, and some 14 ft. of its length is coiled therein; this gives the air feed a high temperature, and effects a saving of oil. The front casting of the furnace has a 2 in. sight hole, fitted with a mica plate, through which the working of the atomisers can be observed. The two flames are set so that they point slightly upwards and converge and meet one another just below the retort, with the result that the force of the jet is broken, and the heat is distributed where most required. A white flame is used for a few minutes at the start until the atomisers and furnace are hot and vaporise the oil properly, after which more air is admitted and less oil, giving a blue flame, with complete combustion. With precipitate carrying only a small percentage of sands or ore slimes, 1,200 lb. can be run down between 6 a.m. and 4 p.m., or 600 lb. per fire per day, this being made up of two pours of about 300 lb. each. The retort will take 150 lb. to 200 lb. precipitate and fluxes on starting, the rest being added as the charge melts down.

The consumption of oil may be reckoned at four gallons per hour for each fire, or one gallon of oil for 15 lb. of roasted precipitate. In May the consumption of oil was 440 gallons, which works out at one gallon per 12·47 lb. of roasted precipitate, but this includes the remelting of bars and mattes, etc., also the melting of two mill bars.

With the oil our smelting occupies some 60 hours per month, as against 140 hours with mixed coke and charcoal on similar material; that is to say, a European smelter and four boys are working six days a month as against 14 days, which it will be allowed, is a very appreciable saving. Add to this advantage the easy control, the regular heat, and the cleanliness of the oil fuel plant, and I think its use will commend itself to those members of the Society whose cyanide plants are located in districts where oil can be obtained at moderate cost. The price of the complete castings for a tilting furnace is about £90 f.o.b. London, and the retorts cost £3 2s. 6d. each, f.o.b. London.

Below is a copy of the log return of two months clean-up.

	MARCH.		MAY.	
	First Clean-up. %	Second Clean-up. %	First Clean-up. %	Second Clean-up. %
Precipitate:				
Gold ...	7·81	7·48	6·67	8·03
Silver ...	48·96	48·96	46·81	48·84
Selenium ...	2·10	3·20	3·86	2·94
Silica ...	10 34	8·74	12·64	10 12
Zinc oxide ...	8 07	7·23	12·35	12·04
Other oxides (as Fe_2O_3)	22 72	24·39	17·67	18·03
Roasted weight, lbs.	1,975	1,567	2,055	1,920
Shorts:				
Gold ...	7·58	6·98	7·35	6·43
Silver ...	47·87	40·25	44·21	43·06
Selenium ...	5·56	4·88	6·06	7·98
Silica ...	10 36	6·80	9·50	10·98
Zinc oxide ...	5·03	13·72	7·62	9·75
Other oxides (as Fe_2O_3)	23·60	27·27	25·26	21·88
Roasted weight, lbs.	526	432	697	815

	MARCH.	MAY.
Precipitate:		
Pressed weight, lbs. ...	6,356	8,294
Roasted weight, lbs. ..	3,542	3,975
Per cent. loss	44·2	52·0
Ounces bullion	32,170	34,416
Ounces gold	3,883	4,207
Ounces silver	26,296	28,090
Ounces bullion per lb. roasted precipitate ...	9·08	8·61
Ounces gold per lb. roasted precipitate ...	1·09	1·05
Ounces silver per lb. roasted precipitate ...	7·42	7·06
Pounds borax per lb. roasted precipitate ...	0·126	0·147
Pounds soda per lb. roasted precipitate ...	0·063	0·070
Pounds slag	1,600	1,740
Slag assay, per ton, gold dwt.	5·6	7·7
Slag assay, per ton, silver dwt.	501·0	1917 2
Shorts:		
Pounds wet weight... ...	3,232	3,600
Pounds pressed weight ...	1,712	2,735
Pounds roasted weight ...	958	1,512
Per cent. loss on wet weight ...	70·3	58·0
Ounces bullion	7,560	12,323
Ounces gold	954	1,676
Ounces silver	5,845	9,638
Ounces bullion per lb. roasted shorts	7·71	8·14
Ounces gold per lb. roasted shorts	1·00	1·17
Ounces silver per lb. roasted shorts	6·10	6·30
Pounds acid per lb. roasted shorts	2·08	1·32
Pounds borax per lb. roasted shorts	0·175	0·148
Pounds soda per lb. roasted shorts	0·058	0·037
Pounds iron per lb. roasted shorts	0·018	0·016
Pounds slag	740	740
Slag assay per ton, gold, dwt.	10·8	32·7
Slag assay per ton, silver, dwt.	1260·2	2785·1
Total weight of slag, lb. ...	2,340	2,480
Average oz. slag per oz. bullion	0·94	0·84
Retorts used	5	4
One retort = lbs. precipitate and shorts	900	1,372
One retort = oz. bullion ...	7,946	11,685
One retort = oz. gold ...	967	1,470
Pounds precipitate and shorts, pressed	8,068	11,029
Pounds precipitate and shorts, roasted	4,500	5,487
Per cent. loss	44·2	50·2
Ounces bullion	39,730	46,739
Ounces gold	4,837	5,883
Ounces silver	32,142	37,728
Ounces bullion per lb. roasted	8·82	8·51
Ounces gold per lb. roasted ...	1·07	1·07
Ounces silver per lb. roasted ...	6·92	6·87
Average fineness, gold ...	121·17	112·3
Average fineness, silver ...	810 12	807·8
Fine metal	931·29	920·1
Oil used, gallons		440·
One gallon oil = lbs. precipitate and shorts ...		12·47
Smelting time, hours ...	140	60

On the 48 analyses of roasted precipitate and shorts made in 1907 the bullion obtained was 5·0% gold and 4·2% silver more than called for.

The President: On your behalf I would like to pass a hearty vote of thanks to Mr. Yates for his valuable contribution. It is certainly interesting to us to receive these notes upon what is done in distant lands such as Sumatra. The oil furnaces there from his remarks appear to be quite a success and to have effected a considerable saving.

THE DISTRIBUTION OF THE GOLD PRODUCED ON THE RAND.

(Read at February Meeting, 1909.)

By T. Kirke Rose, D.Sc., A.R.S.M., M.I.M.M., F.C.S. (Hon. Member).

DISCUSSION.

Mr. W. A. Caldecott (*Past-President*): Our Society is much to be congratulated that its proceedings are enriched by a paper which its author is so peculiarly well fitted to write, and which is so appropriately published in the greatest gold mining centre of the British Empire and of the world. Few men are capable of dealing with a subject so little understood, and still fewer of presenting its intricacies in so lucid and convincing a form. The paper is of such a nature that confident acceptance of the information it contains is our natural attitude, supplemented by an expression of hope that in regard to one or two details the author may still further enlighten us.

There is a great difference between the knowledge available as to some portion of the history of gold bullion in the United States, and in Great Britain. In the former country the mining companies deliver their unrefined bullion to, and receive payment therefor from, the various branches of the United States Mint, and the admirable annual reports of the director of that institution detail most exactly, the methods and costs of the refining processes. This is not the English practice, as the Royal Mint does not refine, and between our mining companies and the mint, there intervenes the private refiner, who taxes the bullion producer at a good deal higher rate than the corresponding governmental charges in the United States.* Possibly Dr. Rose may be good enough to tell us, (*a*) whether it would not be both profitable and advisable for the British Government to revert to their practice of former years, in refining bullion as well as minting coinage, (*b*) to what processes of refining our bullion is at present subjected, before it reaches the Royal Mint, and (*c*) who are the purchasers of the 64·5% which is re-exported? In this connection the existence of "war-chests" of gold coin or bullion in European countries is well-known. Just now great attention is being paid to the relative present and future naval strength of various countries, and some information as to the amounts of the various national reserves of the golden "sinews of war" would be of much interest.

The President: On referring to the table of imports of gold into the United Kingdom, given on p. 266 of the February *Journal*, it will be noticed that the fineness of the bullion gradually improves year by year. What is this due to? Higher up on the same page Dr. Rose gives 875 as the average fineness of South African bullion. The imports from British India and Australasia have altered but little, but South Africa and Rest of World have considerably increased. Now as the fineness given by Dr. Rose for South African bullion is below the average for the whole for any of the given three years; is the increase in the average fineness of the total imports due perhaps to a larger amount of very high grade bullion coming from some part of the Rest of World? If I am correct in my surmise, could the author say from what part this high grade bullion comes?

SOUTH AFRICAN COALS AND THEIR ECONOMICS.

(Read at April Meeting, 1909.)

By Arthur J. Andrews.

DISCUSSION.

Mr. E. J. Laschinger (*Member*): I had intended to prepare a discussion on this paper, for I think it is a paper of great interest. It is certainly of great interest to engineers. There is only one thing I would like to touch upon tonight, and that has reference to the geological formation to which the coal belongs. The author says in the beginning of his paper: that our coal was produced chiefly from leaves and twigs. I think that is wrong, for I know that the Vereeniging coal fields show tremendous casts of tree trunks. There, most of the coal is certainly made by the trunks of large trees which flourished in the coal era. With regard to the amount of dirt and mineral matter present in the coal, he demonstrates that lower level coals are cleaner than higher level coals. Thus Natal coals are cleaner than Middelburg coals, and the Middelburg coals are cleaner than the high veld coals of Johannesburg. I think there is a simple explanation for this. There is, no doubt, in the mind of geologists that the back-bone of the country has been this part of South Africa. This high veld ridge of the Witwatersrand has always been the highest part of South Africa, and we have here the upper end of the coal formation of South Africa. The lower levels have been more favoured with the conditions for the production of good coal. I believe the coal formed on these higher ridges was flooded with masses of sand and mud swept down by mountain streams, and thus became very dirty. In fact, in the coal fields at Vereeniging, in the lower seams you get layers of coal with banket embodied in it, showing that the field had been periodically inundated, and that the layers of vegetation were covered by a mountain torrent which rolled the pebbles along. For the Natal coal we had very much quieter conditions, which gave a chance for the coal vegetation to accumulate quietly, and not be inundated with large masses of sand swept down from the mountains. I think that is the explanation as to why the Natal coals are better than the coals on the higher levels. They have also been subject to greater pressure, and therefore are more anthracitic. This discussion of the subject is merely of scientific or academic interest, but at the same time the more we can learn about the scientific bearings of questions of this kind, the more interest the subject creates ; interest which may also lead to some practical conclusion, or some method of investigation, which will be of great value in practice. I would like to discuss the paper from the practical standpoint, but unfortunately I am not able to do so this evening.

---

* See this *Journal*, vol. vi., 1906, p. 258, and vol. viii., 1908, p. 192.

Prof. J. A. Wilkinson (*Member of Council*) read the following contribution :—

SUGGESTIONS FOR A NEW ATOMIC THEORY.

(*Read at April Meeting, 1909.*)

By James Moir, M.A., D.Sc., F.R.S.S.A. (Vice-President).

DISCUSSION.

Mr. A. Adair (*Member*) : In considering Dr. Moir's paper, one is forcibly reminded, by contrast, of the enormous amount of work done by the chemical giants of the last century to support their ideas; viz., Davy, Berzelius, Gay-Lussac, Dulong and Petit, Dumas, Wöhler and Liebig; H. Kopp, Bunsen and many others. They worked, and then theorised.

The work remains for ever, not so the theories. Dr. Moir proceeds differently. He propounds a new theory, and concocts two new elements, and that without doing a stroke of work, practical work, to support or test his theory and his elements. It is against this practice that my remarks will be directed, partly to save our younger members from being misled, and partly to assist in removing some of the swelling, which has been allowed to accumulate by the worthy doctor. He has done good work already, and I hope in regaining a little commonsense, he will employ his undoubted talents to promote the welfare of our beloved science.

With some 40 years' experience, and on this very suitable occasion, I venture to point out to our budding chemists, that the really important work before us is, to discover the properties of matter, those permanent facts which time does not wither. Also to make use of them.

The opinions, hypotheses, theories, the clumsy attempts to explain chemical reactions or phenomena, are always in the melting pot, always changing, and by comparison with the facts, wholly unreliable and actually of little value. Many of them are foolishly called laws, *e.g.*, Mendelèeff's, on which the author bases his theory. We all use theories, much as insects use their antennæ, to feel ahead, but so seldom do they accord with the ascertained facts, that even the greatest chemists and physicists nearly always fail in their predictions. They, therefore, ascertain the facts first and theorise afterwards, having been duly humbled by their repeated failures to read Nature aright. It is much to be questioned whether hypotheses should be allowed to be published. They often further obscure that hidden pathway towards the truth, which we are conscious of, but which is so difficult to find.

There cannot be any question about those mischievous theories which, like the author's, are unsupported by experimental data. They are weeds. Bunsen, the celebrated chemist, was accustomed to say of them in his courteous way. "This is pure theory without any foundation, I will therefore pass it without any further remark." Nor can the author's theory be taken seriously, it is absurd—delightfully absurd. Think of his "spiders," and of his method of manufacturing new elements.

It cost the Curies, I understand, some thousands of pounds, and years of work, to discover radium, and study its properties. The doctor will not take the trouble to discover new elements, he makes them ! It may be disappointing, but how easy now to get a new element, when it is wanted. The method is, of course, mathematical. Thus, you write an equation 12 over $4 = 3$. 3 is a new element. The author names it Zoîkon, symbol z. It has no body and no properties—it is all soul. He then makes x, for which the equation is $O + O = 2$. This body he hesitates to name, perhaps because he has given it negative virtues. It is found in a circle, crossed. It makes one gloomy to be told that this new born element has been *already* predic(h)ated (sic). One would think that the symbol would protect it from predicts or other malign influences.

I must, however, show the doctor the crudity of his sub-constitutional formulæ applied to the metals. One example will suffice. Let us take the alkali metals. It cannot be denied that they are electro-positive, and very much so. Taste a little sodium if you wish to be powerfully convinced. Now look at the doctor's formulæ and you will find that he has actually made them *negative*! Thus, the new gases are neutral, *z* has not got any properties. So the double-armed, more numerous negative *x's* completely overpower the single-armed, positive H's.

Mr. Watson has already shown the absurdity of the doctor's theory, when it is applied to organic chemistry.

Prof. Wilkinson gives an example of the interesting results obtainable by judiciously arranging the atomic weights, the game all of us played at as students, and very fascinating it was.

The preceding was written before I received the May number of the *Journal*, containing Mr. Roux's contribution to the discussion, which opportunely enough, confirms my remarks about mischievous theories. As Mr. Roux is a visitor, one has to remember the rights of hospitality, but one would hardly be human not to appreciate that exquisite little sentence about the seven, also not to express the hope that he will not involve us in any new chemical laws until time crowns him with grey hairs or their equivalent.

STOPE MEASUREMENTS.

(Read at May Meeting, 1909.)

By Otto S. Tonnesen (Member).

DISCUSSION.

The President: I see none of our members are prepared to discuss Mr. Tonnesen's paper this evening. It was necessary to have the diagrams and illustrations before one before it could be properly understood. I feel sure that many of our surveyor members will be coming forward later on with quite a lot of discussion on it.

Mr. O. S. Tonnesen (*Member*): I promised to give some results of my experiments for the last month. They have proved satisfactory inasmuch as I have been able to measure in one stope, alone, better than I could formerly do with assistants.

RESEARCHES UPON THE TELLURIDE GOLD ORES OF CRIPPLE CREEK (COLORADO).

(Read at May Meeting, 1909.)

By the Portland Metallurgical Society
(edited by Thos. B. Crowe).

DISCUSSION.

Mr. W. A. Caldecott (*Past-President*): Discussion upon this paper would most appropriately come from those engaged in the treatment of telluride gold ores, and it is to be hoped that some of our members with Westralian experience will respond to the author's appeal, and thus once more evidence the world-wide co-operative spirit of our Society.

I had the pleasure in 1907 of going over the Portland mill and, although a stranger, of receiving a most courteous reception and ready explanation of the methods then employed. The superintendent, Mr. G. M. Taylor, is now one of our Corresponding Members of Council, and his genial enthusiasm is displayed in the authorship title of this paper, in that the "Portland Metallurgical Society" is an informal semi-social gathering fortnightly, of the staff, to discuss the problems with which they have to deal.

As stated by the author, mining companies in his district are confronted by the fact, that the low-grade telluride ore will not pay the cost of transport, plus roasting, chlorination, and concentration.* With us here, fortunately, methods already developed are suitable for lower-grade ores, but the present relatively low working costs on telluride ores at Kalgurli, have only been evolved at the cost of much money, time, and labour, and it is hence from the latter field rather than from our own, that valuable information is most likely to be available for Cripple Creek. Since the complex ores, with which the author is dealing, are said to contain sulphides of copper, lead, zinc and antimony, it would assist in the solution of the problem to publish a complete quantitative analysis of a typical sample, the minerals present being also identified.

The paper states that the difficulty in practice of using as oxidisers, persulphates and hypoiodites, is their reduction by pyrite: to this disability other oxidisers, such as potassium permanganate, are also subject, though barium dioxide is probably less so than most. A useful list of possible oxidisers for cyanide treatment has been published by Mr. D. Mosher in the *Mining and Scientific Press* for May 15, 1909.† This table includes ozone, chlorine (electrolytic) and chlorine peroxide, and gives the cost and percentage of available oxygen in each.

In its essence, the reluctance with which gold combined with tellurium dissolves, seems obviously due to the fact, that one gold compound (telluride) must be decomposed before another (potassium auro-cyanide) can be formed, whereas with metallic gold the first stage of the process is not required, and oxygen present in solution sets free, by combining with potassium in KCy, the cyanogen radical in a nascent state to combine with the metallic gold. Roasting, of course, breaks up the gold-tellurium compound, but what appears to be required under the conditions outlined by the author, is, to find some soluble reagent, which is compatible with cyanide in solution, is unaffected by pyrite, and has a greater affinity for tellurium than has gold, so that the latter may be set free to dissolve as a double cyanide.

---

* See this *Journal*, p. 438,

† See this *Journal*, p. 437,

On the motion of Mr. White it was resolved that coffee should be provided at the meetings of the Society in future.

The President then declared the meeting closed.

Contributions and Correspondence.

ROUTINE ASSAYING ON A WESTRALIAN MINE.

I would like to reply to Mr. White's quick though ill thought out reply, to a small contribution of mine on Mr. Blyth's paper.

Mr. White has done the cause of the use of borax in fluxing, a decidedly good turn, in bringing forward this quotation from some recent work of Mr. Clennell. For instance, (1) "The use of a large excess of borax has a tendency, etc." The most ardent supporters of the use of borax, have always cried out against its use in excess, and merely claim its use in moderation. In paragraph 3 we have (3) "The presence of borax makes the charge distinctly more fusible and gives, usually, a clear greenish slag. By using a small quantity of borax in the charge, not exceeding 5 gm. to 10 gm. per A.T. of ore, it was found that a uniform fusible slag was produced without any noticeable amount of scum, and that the buttons could be detached from the slag by a slight blow, without any portion of lead adhering." Later on, "taking the results as a whole, it may be said that they show no marked advantage either way, and they lend no support to the idea, that borax carries any appreciable quantity of gold into the slag . . . . . . The addition of a small amount of borax seems, however, to be a slight advantage, perhaps chiefly on account of the greater fusibility which it imparts to the slag."

In the face of the above quotations from the paper, which seem to support borax flux. he refers to the "small difference in favour of *no borax*."

Quotations, like boomerangs in the hands of an unskilful thrower, are apt to come back, and lie where they are least expected."

My extreme variations, which Mr. White refers to, I am at a loss to understand, particularly any connection with the use of borax; I am of course, referring to the variations when using magnesia cupels, as stated at the end of my last contribution. Borax, I might remind Mr. White, is never used in cupellation, even by its most ardent supporters For the usual comparison of different cupels I make up portions of check gold and test lead in order to keep a uniformity of button, and do not rely upon buttons obtained by crucible fusions which may vary slightly in composition.

As to the puzzle, "find the missing ingredient." The query should be answered by Mr. White, not by myself, for if he refers to vol. viii., No. 5, p. 151, of our Transactions, he will see the charge which gave such queer results, and the one I so strongly condemned is the one he advised me to use, as it was sure to give excellent results. As Mr. White does not care for the use of borax, the puzzle is certainly for him, and not for me, to answer, but personally I would advise him to use the much belaboured "hoary headed superstition."

L. J. Wilmoth.

Simmer and Jack Proprietary Mines, Ltd.,
June 21st, 1909.

Notices and Abstracts of Articles and Papers.

CHEMISTRY.

Silver Cupellation Temperatures.—"Prof. Chas. H. Fulton and pupils conclude from an investigation upon this subject that: (1) In case of the leads buttons, not containing any appreciable amount of copper ore iron, etc., *i.e.*, pure, that a muffle temperature of at least 800° C., and, better, one of 850°, is necessary to 'uncover' or start cupellation ; (2) that this temperature may be lowered to about 770° C., during the oxidation of the greater part of the lead ; (3) that toward the end of the cupellation, or the 'finishing,' in case of silver, it must again be raised to about 830° C., in order to get a pure silver button ; (4) that the actual temperature of the cupelling lead is always appreciably higher than the muffle temperature ; (5) that the actual finishing temperature of the cupellation cannot safely be carried below about 910° C. ; (6) that the greatest observed surfusion of silver was 77° C., and that this is probably very near the maximum ; (7) that silver beads finishing with surfusion are free from lead ; (8) that 'feathers' or crystals of sublimed litharge on the cupel are an indication of the proper cupellation temperature, provided the air draft is not excessive."—Prof Chas. H. Fulton.—*Chemical Engineer*, Nov., 1908, p. 215. (A. McA. J.)

Corrosion of Iron.—"An important contribution on the corrosion of iron was contained in a paper recently presented before the Boston (U.S.A.) Society of Arts by Dr. William H. Walker, director of the research laboratory of applied chemistry of the Massachusetts Institute of Technology. The electrolytic theory of corrosion, which has been so ably advanced by Dr. A. S. Cushman, of Washington, D.C., was first explained, and the method of detecting its existence with a solution of potassium ferricyanide to disclose the presence of iron ions by the bright blue colour formed, and the presence of hydroxyl ions with phenol phthalein shown by red colour. Impurities in iron accelerated corrosion by establishing electric currents due to difference of potential between different spots; hence the reason for the slower rusting of the purer iron or steel. It was possible now to buy iron on the open market containing not over 0·00005 total impurities. Even slightly acidulated water promoted corrosion because of the greater number of hydrogen ions present. Conversely the corrosion was decreased by rendering the water alkaline. For example, iron would not rust in a solution of soda ash, limewater, or sodium phosphate, a fact which answered the question whether or not concrete protected iron. In setting, Portland cement liberated a large quantity of caustic lime which prevented rusting, but if the concrete was not impervious water percolating through would carry away the easily soluble calcium hydrate, and rusting could then proceed. This explained the rusting which had been discovered in the Charles River bridge at Boston. Water from the highway had leaked down to the concrete and between the joints in the supporting I beams, washing out the calcium hydrate. This strongly alkaline solution had saponified the linseed oil paint, and, subsequently removed by the rain, exposed the unprotected iron to rusting. Corrosion was also accelerated when there was connection between iron and a metal such as copper, lead, or bronze. It suggested a means of protecting the latter, but to protect iron a more positive metal such as zinc should be connected with iron. This explained the use of zinc in protecting boiler tubes from pitting when there was an electrical connection. In perfectly pure water zinc would protect iron for a distance of only about ½ in. Salt or other electrolyte added to the water would increase the area of protection to 20 in. or more. Mill scale (magnetic oxide of iron) was a substance on which hydrogen could be liberated more easily than on iron; hence where there were breaks in this scale, corrosion would ensue, as in the pitting of boiler tubes. It was a safeguard to remove all of this scale from all kinds of steelwork. Rusting could not continue in the absence of oxygen; therefore boiler steel or tubing corrosion could be prevented by removing the air from the feed water before it was admitted to the boiler. This might be done by using an open feed water heater or connecting the feed heater to the dry vacuum pump of the condenser, or the water might be drawn through a closed box containing scrap iron where the oxygen in the water would be used up in corroding the scrap iron, instead of the boiler tubes. The oxygen in the water might also be absorbed by feeding with the water a small quantity of an alkaline solution of a tannin material, whereupon a pyrogallate of the alkali would be formed, rapidly absorbing the oxygen. Soda ash or other alkali was useful not because it affected the oxygen content, but because it decreased the hydrogen concentration, as before mentioned. Tin, like copper, aided the oxidation of the hydrogen set free by the reaction. Imperfections or pin-holes in the tin coating of sheet iron therefore invited corrosion. To detect these imperfections a solution of gelatine or glue containing dissolved potassium ferricyanide might be flowed over the tin plate, and wherever bright blue spots appeared there were holes through the coating, exposing the iron."—W. H. WALKER.—*Times Engineering Supplement*, March 10, 1909. (J. A. W.)

SILUNDUM.—"Extensive researches on the use of silicon carbide for electrical resistances to stand high temperatures have resulted in the discovery by Herr F. Bölling, a German Engineer, of a process for converting any piece of carbon wholly or partially into silicon carbide. In Herr Bölling's early experiments he attempted to produce resistances by mixing silicon carbide powder with bonds of various compositions to bind the individual crystals together, but he found that resistances made in this way, though non-conductors at low temperatures became conducting at about 700° to 800° C. Later, however, it was discovered that if carbon and silicon be heated together out of contact with air to a temperature exceeding 1,600° C., the point at which silicon vapourises, the silicon will penetrate into the carbon, which keeps its shape, and form with it a layer of silicon carbide of any desired thickness. In this way it is possible to produce resistances without binding materials, which will stand temperatures as high as 1,650° to 1,700° C. The silicified carbon obtained by this process, to which the name 'silundum' has been given, is now produced commercially in Europe at a plant in Switzerland especially erected for this purpose.

For the production of this material, electric furnaces are used, of the same kind as for the manufacture of carborundum. The pieces of carbon of the various shapes are introduced into a mixture of sand and carbon, which is acted upon by a granular core of coke as conductor for the electric current. Instead of sand and carbon, amorphous carborundum can be used, enriched with sand, to provide the necessary silicon for the carbon pieces to be converted into silundum.

The silicification of the carbon depends on the time and the temperature employed. It is possible to entirely silicify the carbon pieces or to cover them only with a layer of silundum.

If it is desired that one part of the carbon piece remains carbon on one spot, this place is shielded by another piece of carbon, thus preventing the silicon from penetrating into that particular spot. Lamp filaments can be easily converted into silundum, or only covered with a layer of silicon carbide. Charcoal is very easily converted into silundum; so is coke, because both these materials are very porous.

Graphite is easily silicified. Carbon powder compressed at very high pressure can be silicified without being baked first.

Silundum is another form of silicon carbide and it has generally the same properties. It is very hard; it resists temperatures up to about 1,600° C. when heated in air and does not oxidise. It resists the attack of acids when cold, and is a conductor of electricity, its resistance being several times that of carbon.

The hardness of silundum is variable, and depends on the zone around the core of the furnace in which it is produced. Material from the amorphous zone, is less hard than that obtained in the crystalline zone. It may be assumed that the hardness depends on the temperature in which the material is produced. The hardness may also depend on the amount of silicon taken up by the carbon. It seems that some pieces contain more or less silicon than others, but no analysis was made to ascertain this.

In regard to the fireproof qualities of silundum, it may be mentioned that at about 1,700° C. the silicon leaves the carbon and combines with the oxygen of the air. Small transparent globules are formed (for instance, on a rod that is put in a circuit and heated to about 1,700 by the electric current) on the surface, of clear or brownish colour. These globules are silicon dioxide. Parts of the silicon escape in form of a white vapour.

Silundum cannot be melted, and it behaves in this respect like carbon. The electric resistance of silundum is variable, and depends on the kind of carbon and its hardness. Silundum made from porous carbon has a higher resistance than when made of hard carbon. The resistance depends also on the modification of the material. If produced in the amorphous zone, the resistance is generally higher than in the crystalline zone.

The first use for which the material found application was for electric cooking and heating purposes. Silundum replaces platinum for this purpose and is much cheaper.

Silundum stands very high heat, does not oxidise, and can be used with the necessary high temperature needed for the purpose. Ranges are produced that have the glowing heat of a coal fire and they are used exactly as those heated by coal.

For industrial purposes there is a large field for heating apparatus for melting metals, brass, aluminium, lead, tin, etc., and for electric laboratory ovens for high temperatures, and it is especially for this purpose that the material fills a long-felt want. Platinum and platinum-iridium has been used to a great extent for laboratory ovens, but such apparatus are very expensive and they can be repaired only at great expense. Silundum can be used with just as high temperatures as platinum and its price is low. The first cost of the apparatus as well as the expense for repairs are considerably lower than with platinum. For heating purposes the silundum rods can be used single in lengths up to 800 mm., depending, of course, on the diameter, as solid, round, flat or square rods or tubes or made into grids, mounted in a frame and provided with contact wires.

Silundum stands just as well as silicon carbide the attack of acids, and experiments are now being made to use it as electrodes for electric bleaching purposes, instead of platinum-iridium, carbon or graphite. In order to prevent the oxidation of carbon electrodes used for the manufacture of ferrosilicon, it has been proposed to cover these electrodes with a layer of silundum.

Silundum can be nickel plated or covered with a layer of platinum. It is a very refractory material and does not melt. It is, however, attacked at high temperatures by molten metals, but this should not prevent its use as fireproof material, for it is easily possible to cover the outside of a carbon or graphite crucible or the like with silundum and leave the inside carbon. It is not necessary to silicify pieces to be used for fireproof purposes entirely, but only to a certain depth and according to the use for which they are to be employed. Carbon bricks or any shaped pieces can be covered with a layer of silundum at small cost. It is thus possible to obtain a material which for many purposes will be far superior to any fireproof material so far known.

Silicified coke can be used instead of ferrosilicon or carborundum in the steel industries, and, as it can be produced at very low cost, it will doubtless find extended application."—F. Bolling, *Electrochemical and Metallurgical Industry*, Jan. 1909.—*Engineering Magazine*, March, 1909, p. 1018. (J. A. W.)

Sulphur and Corrosion in Steel. —"This paper was read by Mr. G. N. Huntly, who had examined a case of pitting in a boiler. It was found that each pit was the centre of a blister. The liquid inside the blister differed entirely in composition from the boiler water, the former consisting of a slightly acid solution of ferrous sulphate, whilst the boiler water was alkaline with caustic soda. The solid matter in the envelope of the blister, which consisted of a mixture of iron oxides, acted as a semi-permeable membrane keeping apart the ferrous sulphate within and the caustic soda outside the blister. Reasons were given for supposing that the particles of manganese sulphide, segregated in the steel, were oxidised by the oxygen dissolved in the boiler water to sulphuric acid and an oxide of manganese, the acid then acting locally on the surface of the boiler-plate in the vicinity of the particles of manganese sulphide. This pitting of the boiler-plate was prevented by adding sodium arsenite to the boiler water, the efficacy of this reagent being probably due to its power of taking up the dissolved oxygen.—G. N. Huntly, *Times Engineering Supplement*, March 3, 1909.—*Society of Chemical Industry*, March 1, 1909. (J. A. W.)

METALLURGY.

Cyanidation of Silver Ores.—*Table Showing Costs of Oxidising Agents.*

Chemical Product.	Available oxygen	Costs per lb.	Cost, efficient oxygen per lb.	Remarks.
Sodium di-oxide Na_2O_2	20·5	1/8d.	8/4d.	
Barium di-oxide BaO_2	8·0	4d.	4/2d.	
Hydrogen peroxide H_2O_2 = 3%	1·5	6d.	33/4d.	
Potassium permanganate $K_2Mn_2O_8$	12·7	5d.	3/4d.	
Sodium bi-chromate $Na_2Cr_2O_7$	17·9	3½d.	1/7½d.	
Calcium hypochlorite $CaOCl_2$	7·4	⅝d.	8½d.	
Potassium ferri-cyanide $K_6(Cn)_{12}Fe_2$	—	1/5½d.	—	Indirect agent in oxidation in presence of alkali.
Ozone (O_3)	—	5d.	4/7d.	8 lb. per kw. day in form of ozonised air.
Chlorine (Electrolytic)	—	2d.	—	Not crediting caustic soda produced at the cathode.
Ammonium persulphate $(NH_4)_2S_2O_8$	7·0	2/6d.	35/-	
"Linde" oxygen	96-98	1/3d.	1/3d.	With "Linde" apparatus cost of 96% to 98% will be 1d. per pound.
Chlorine peroxide Cl_2O_2	31	1/5½d.	4/9d.	

—D. Mosher.—*Mining and Scientific Press*, May 15, 1909. (W. A. C.)

Cyaniding Cripple Creek Dumps. — "For several years past Colorado metallurgists have been attempting to solve the metallurgical puzzle presented by the low-grade ores of Cripple Creek. Millons of tons of ore running from $3 to $10 per ton have lain on the great dumps for years, while other millions were exposed in hundreds of miles of underground workings. The Portland Company alone had enough low-grade ore in its 40 odd miles of workings to materially increase the Cripple Creek gold production provided a method of profitable treatment could be devised. Shipping the ore to the Colorado Springs reduction plant of the company cost $1 per ton, roasting the ore cost $·76 per ton, and the other milling expenses ate up the small margin of profit. Thus, for some years previous to 1907, the issues were clearly presented, first, to eliminate roasting the ores if possible, and, second, to eliminate freight charges by building mills at the mines.

It will, of course, be understood that until a way of eliminating the roasting of the tellurium ores was discovered, it would be useless to erect mills at the Cripple Creek mine, where roasting costs even more than at Colorado Springs.

The first great plant to be erected at the mines was that of the Stratton Independence Co. nearly two years ago. Little information is available concerning this, except that the method of treatment followed was fine grinding of ore, followed by cyaniding. Owing to the low grade of this company's dump ores the mill has not so readily proved successful up to the present time as it might had it treated the dumps of other mines.

In the meantime, the great Golden Cycle plant at Colorado City has been completed and successfully operated by roasting the ores and then cyaniding. During the year 1908 this plant is said to have produced roughly $6,000,000 in gold. Owing to the fact, it is commonly believed, that the cyaniding of low-grade ores at this plant gave a chance for little or no profit, the company has lately decided to erect a plant at its Cripple Creek mine. Ground has been broken for the erection of this plant and construction work is progressing rapidly.

Contrary to the conclusions of the Stratton Independence and Portland Companies' metallurgists, those of the Golden Cycle are to the effect that concentration of the low-grade ores before shipment to the Colorada Springs mills will solve the problem. According to Manager McGarry, the exhaustive tests carried on at the company's experimental plant at Colorado City have resulted in a saving of 65% of the gross value of these low-grade ores. It is, therefore, planned to treat those ores which will not stand transportation charges, by a system of dry concentration at a ratio of 6 tons crude ore into 1 of concentrates. These concentrates will then be shipped to Colorado City for treatment at the company's cyanide plant. The mill in the course of construction on Bull Hill, near the Golden Cycle's main shaft, is to have the initial capacity of 80 tons of concentrates per 24 hours.

During the two years in which the Stratton Independence and Golden Cycle metallurgists have been evolving their plants, those of the Portland Company have also been most active. It is now common knowledge that the widely advertised Moore-Clancy process, was given an extended trial at the experimental mill of the company but was evidently found unsuitable. The great amount of experimental work, however, did result in the evolution of what is known as the 'Portland process,' the details of which are as yet unavailable. It has been thoroughly tried at the company's Cripple Creek 10-stamp experimental plant however, with such success that a large mill is soon to be built close to the mines. The annual report of the company, recently issued, says that from many tests the company is convinced that the problem of treating the low-grade dump ores has been solved. The directors have decided to back up their judgment by the expenditure of at least $100,000 for a 300-ton mill, erected near the mine on Battle Mountain, to treat the great company dumps. These contain enough ore to run the mill for many years at an estimated net profit of from $1 to $2·50 per ton.

The building of these new mills is expected to have a regenerating effect on the entire camp of Cripple Creek, which, combined with the events expected to follow close upon the completion of the Roosevelt drainage tunnel, is likely to prolong the life of the great camp for many years."—*Mines and Minerals*, May, 1909, p. 444. (W. A. C.)

THE ECONOMICS OF AMALGAMATION.—"While the great rise in the cyanide process for treating gold ores has cast more or less disfavour on mills depending primarily on amalgamation for their gold and silver extractions, yet amalgamation still extracts more value in gold and silver than any other milling process. Amalgamation is still given the preference wherever possible. The tailings may be cyanided, concentrated, etc., but still the largest percentage of values comes from the plates. In the 'Mineral Resources of the United States for 1906,' published by the United States Geological Survey, it is estimated that 63% of the gold produced, comes from dry or silicious ores. Probably 60% of this is produced by amalgamation. There is very little of this ore smelted and only a few cases, notably in Colorado and Arizona, where it is concentrated without amalgamation; also comparatively few mills where cyaniding is carried on without amalgamation at one point or another. Silver ores are more complicated, as only 29 3% of the total production is estimated as obtained from silicious ores. Of this, there is probably not over 40% to 50% obtained by amalgamation.

The above only suggests the importance of amalgamation to gold and silver production, and yet very little is written on this subject.

Henry Louis, in his book on gold milling, says: 'All that is necessary is for clean gold to come in contact with clean mercury.' There we have the whole idea in a nutshell, although it sounds simpler than it is.

Taking the above statement as a basis we must first have our gold as clean as conditions permit; second, arrange the amalgamation apparatus to bring the greatest quantity of gold in contact with mercury, and, time and space being important, to this may be added 'in the shortest time.' I will only say in regard to the first condition that we usually have to take the ore as we find it. If it is clean enough, all right. If not, we may help matters by the conservative use of such chemicals as potassium cyanide, sodium amalgam, lye, etc., depending on the ore. Also high battery discharge may prove of value for a scouring action—but rarely.

Proceeding to the second condition, we find a number of ways to control, or cause, more or less contact to take place. We may feel sure that with the ordinary plates operated in almost any old way, we will get practically all the coarse gold to amalgamate. As usual in milling practice, it is the elusive little slime value that causes the expenditure of 'gray matter.'

Starting on the hypothesis that if we can catch the fine gold, the coarse will take care of itself, let us see by what means this can be done. We will take the ordinary amalgamation plate, and to simplify matters, we will say in one piece, and not stepped. The important point now is, to cause as much heavy gold or silver particles to settle on to the mercury as possible. The means of doing this which we first think of are slope, rate of flow of the pulp, dilution of the pulp, and in some cases, vibration. These are difficult to explain, and only by watching and working with properly adjusted plates can these points be learned. It might be well to mention, however, that these are all dependable on each other and are adjusted relative to each other so as to fulfil the following conditions: The pulp must be as thin as the proper working of the battery will permit. The flow of the pulp also is limited by the working of the battery. This in turn, limits the slope of the table as, obviously, the table must have slope enough to remain clear of sand. The best conditions for allowing the fine gold to settle evidently are as little slope as possible, as thin a pulp and as small a rate of flow as possible, and as these are all in turn dependent on the proper working of the battery, we do not get much leeway in the matter of adjustment.

In some cases, vibration of the table is resorted to, and in general may be said to largely increase the amount of gold caught. This, however, invariably causes an increase in working expenses both for power and labour, and the ledger is the final judge of its practicability. Vibration has a tendency to cause extremely hard amalgam to form, and is usually condemned for that reason. The use, though, of amalgamating plates on bumping tables, feed-boards of vanners, etc., has at times been accompanied by a paying increased extraction.

The points so far mentioned, however, are discussed thoroughly in many textbooks and the object of this paper is to call attention to one point of adjustment that is largely overlooked. That is, the condition of the amalgam on the plate. This in turn leads to a discussion of the proper time to dress, sprinkle, and clean up.

The operation of settling heavy particles of ore on a non-shaking table is studied closer in canvas plants than in amalgamating. Here one rule forces itself irresistibly to the front; *i.e.*, the longer the nap, other conditions being the same, the more fine particles of concentrates are caught. This also is true in amalgamating. Therefore the best results are obtained when the amalgam forms a rough and almost fibrous nap on the plates, at the same time having a slight excess of quick.

When plates have just been cleaned and dressed, the amalgam is almost perfectly smooth. To offset this to a small extent, the brushes are moved back and forth across the plates, thus forming riffles which temporarily take the place of the nap until it can begin to form. Now the pulp begins to flow over the plate on which is a large excess of 'quick.' The coarser gold immediately sinks into the bath of mercury. Gradually a thickening of the quick takes place and a long matted nap begins to form. As this continues, more and more fine gold is entangled —gold that was swept clear of the mercury when the only roughness was riffles caused by the brushes. This action continues until the amalgam becomes too hard, or from lack of 'quick' a crust forms on the surface. In the 'too hard' condition, a little 'quick' sprinkled on will again start the action, in which case, the broom or brush is not touched to the plate, as that would instantly destroy the fine nap so carefully formed. In case a crust has formed, such that the drops of mercury thrown on simply roll off, and are not instantly amalgamated, then dressing with the brush must be resorted to. In this case then the nap must begin to form again from the start.

To illustrate this more fully, a reference to the plotting Fig. I. will be in order. This was constructed from data obtained while superintendent of the Tucker mill, in Gilpin County, where we cleaned up once a day. Every hour a sample was taken from the lip of the mortar and one from the plate tails. These were assayed with the extraction for that moment calculated. The curve starts immediately after the clean-up. The solid line shows the plotting of the extractions so obtained. In this case, the plates were examined each hour immediately following the taking of the samples, and if necessary 'quick' was thrown on. As a matter of fact, no 'quick' was necessary until after the eighth hour, after that it was added each hour.

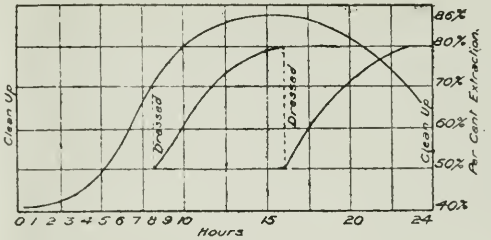


FIG. I.

From this curve, the reasoning above is made clear. The clean-up should of course take place at the time necessary to get the maximum extraction. The dotted line shows a similar plotting in which two dressings during the twenty-four hours are shown. These dressings were not necessary as the plates were not crusted and the curve shows it would have been bad policy to continue to dress them.

All plates when plotted in this manner do not show a rising line to start with. One set of plates of which I plotted the line, began to drop immediately after dressing, that is the maximum extraction was obtained at the very first. We studied this very carefully and decided that it was caused by almost the entire absence of fine gold and the fact that very little mercury was used. Under these conditions, the first coarse gold would immediately sink; as this continued, more and more gold escaped, probably on account of too little 'quick.' What nap formed, would be of little value, as all the gold caught by the nap would be washed off with the hose; being mechanically held, and not amalgamated. Thus the plates would become mere concentrators. In this case, the plotting would indicate the use of more 'quick' and clean-ups as often as possible. This was done and the effect was to raise the extraction on the plates from about 56% to over 60%.

The newer scheme of combining the cyanide process with amalgamation by stamping in a cyanide solution gives altogether new problems to face. In the first place, the cyanide softens the mercury and amalgam so that it flows readily, and is hard to retain on the plates. This flow tends to harden the plates in spots. In this way it has the same effect as though the amalgamation was being carried on in hot water. Another point which has to be considered here is, that the cyanide is continually dissolving gold out of the amalgam. Louis proved that gold combined in amalgam is much more readily attacked by cyanide than free gold. Thus there are two opposing actions going on continually; the 'quick' taking free gold from the ore and the cyanide extracting gold from the amalgam.

It is said that in this case the line falls sharply from the start. Here it cannot be lack of mercury, as a large excess was used. Under these circumstances might not the action of the cyanide on the amalgam explain it? The first gold amalgamated after starting would sink in a large excess of 'quick.'

In this way, the action of cyanide would be reduced to a minimum. As more and more gold is caught, the inevitable nap begins to form. The more it forms, the more the cyanide attacks the gold in it, consequently the faster the extraction falls, following simply the laws of mass action: but the action of cyanide is slow, and that of amalgamation is comparatively fast. Therefore the amalgam increases on the plates, but the percentage of extraction falls from the start.

The above facts, together with the probability that the cyanide slowly prepares gold for amalgamation that might otherwise escape, will explain to a certain extent, the fact that the amalgam, where the ore is crushed in cyanide solution, forms the entire length of plate.

In most mills, the actual portion of the plates covered with amalgam is confined to the upper three quarters. In fact if amalgam forms clear to the lower edge, it is usually said the mill has insufficient plate area. At the Liberty Bell mill, Telluride, where the ore is crushed in cyanide, there are 16 ft. of plates used and the amalgam is almost as heavy at the last end as it is under the lip."—JUSTIN H. HAYNES.—*Mines and Minerals*, Feb., 1909, p. 321. (A. R.)

COMPETITOR OF THE ELECTRIC FURNACE.—"Experiments by Mr. Baraduc Muller demonstrate that carborundum or silicon carbide reacts on metallic oxides energetically, like powdered aluminium in the thermit process, and at a relatively low temperature —about 1,300° C. Consequently, alloys can be manufactured without an electric furnace—viz., ferro-silicon, copper, nickel, chromium, manganese, etc.—perfectly similar in composition to those from the electric furnace. Anyone, therefore, can thus obtain silicides, with a luted crucible, in any quantity, from a small to a large scale.—*Metaux et Alliages.—London Mining Journal*, March 6, 1909, p. 306. (A. R.)

THE CARRYING CAPACITY OF LAUNDERS.—"The data for the following paper are taken from a portion of the experimental work on 'The Carrying Capacity of Launders,' which has been conducted at the Utah State School of Mines, during the past three years. This work has been under the direction of Prof. G. A. Overstrom, head of the department of mining and milling, and covers about 3,000 experiments. It has been the aim to secure such results as would enable the millman to calculate, or at least get some idea of the carrying capacity of a wooden launder, with different sized materials, to be carried by different amounts of water, and on the various grades that are used in mill practice.

A set of experiments has also been made by Mr. S. M. Parmley and the writer, during the past year, to show a comparison of the capacities of wood-, glass-, and linoleum-bottomed launders of the same size and under like conditions.

Material.—The materials used in these experiments were 40—60 mesh pyrite, 10—30 mesh tailings, and 4—8 mesh gravel. The tailings were taken from the results of a mill test. They were almost pure limestone, containing a small fraction of a per cent. galena. They ran about 40% 10—20 mesh and 60% 20—30 mesh material. The 40—60 mesh pyrite was very clean concentrates from the Overstrom and Wilfley tables. The gravel was ordinary rounded pebbles of 4—8 mesh.

Our purpose in taking the above materials was to secure difference in size, structure, and composition. A difference in structure (*i.e.*, rounded or angular) makes a big difference in capacity, as will be shown later.

10 LB. 40—60 MESH PYRITE.
$2\frac{1}{2}$ in. Wood Launder.

Table 1*a*.

Grade.	10 lb. of water.		20 lb. of water.		40 lb. of water.		60 lb. of water.	
	Time.	Capacity per sec of 1 lb of water.	Time.	Capacity per sec. of 1 lb, of water.	Time.	Capacity per sec of 1 lb, of water.	Time.	Capacity per sec of 1 lb of water.
in.	h. m. sec.		h. m. sec.		m. sec.		min. sec.	
$\frac{1}{4}$			1 45 44	·001891	22 35	·004422	10 36	·00646
$\frac{3}{8}$	1 20 14	·00496	21 35	·00927	7 05	·01410	5 19	·01254
$\frac{1}{2}$	35 40	·01118	9 55	·01955	4 08	·02418	3 15	·0205
$\frac{3}{4}$	12 56	·0309	4 18	·04640	2 03	·04878	1 $39\frac{1}{2}$	·0463
1	6 58	·0576	2 46	.0722	1 15	·0801	1 00	·0667
$1\frac{1}{4}$	4 26	·0903	1 52	·1071	0 56	·1071	0 $41\frac{1}{2}$	·0964
$1\frac{1}{2}$	3 0	·1335	1 28	·1362	0 38	·1578	0 $29\frac{1}{2}$	·1355
2	1 43	·2330	0 $47\frac{1}{2}$	·2525	0 26	·2110	0 $16\frac{1}{2}$	·2425
$2\frac{1}{2}$	1 06	·3640	0 30	·3990	0 17	·3522	0 $9\frac{1}{2}$	·4220

$2\frac{1}{2}$ in. Glass Launder.

Table 1*b*.

$\frac{1}{4}$					17 45	·00564	11 30	·0058
$\frac{3}{8}$			26 23	·00757	6 46	·01475	5 19	·01254
$\frac{1}{2}$	36 47	·01085	11 26	·01745	3 $54\frac{1}{2}$	·02558	3 12	·02082
$\frac{3}{4}$	12 43	·03160	4 15	·03265	2 $00\frac{1}{2}$	·04960	1 37	·0412
1	6 59	·0574	2 45	·07260	1 14	·08120	0 $58\frac{1}{2}$	·0684
$1\frac{1}{4}$	4 15	·0941	1 54	·1051	0 54	·1112	0 40	·1000
$1\frac{1}{2}$	2 57	·1235	1 25	·1461	0 $39\frac{1}{2}$	·1518	0 $27\frac{1}{2}$	·1454
2	1 42	·2350	0 46	·2650	0 23	·2610	0 16	·2500
$2\frac{1}{2}$	1 07	·3580	0 27	·4460	0 16	·3750	0 09	·4450

Table 1*c*.
40—60 Mesh Pyrite. $2\frac{1}{2}$ in. Linoleum Launder.

Grade.	5 lb. of material. Time.	Capacity per sec. of 1 lb of water	5 lb. of material. Time.	Capacity per sec. of 1 lb. of water.				
in.	min. sec.		min. sec.					
$\frac{1}{4}$			29 30	·00339				
$\frac{3}{8}$			9 13	·01080				
$\frac{1}{2}$	12 36	·01588	5 4	·0197				
$\frac{3}{4}$	5 16	·03795	2 14	·04475				
1	3 03	·0655	1 $20\frac{1}{2}$	·0745				
$1\frac{1}{4}$	2 $03\frac{1}{4}$	·0973	0 58	·1035				
$1\frac{1}{2}$	1 $27\frac{1}{2}$	·1370	0 $40\frac{1}{4}$	·1480				
2	0 $50\frac{1}{4}$	·2390	0 $30\frac{1}{4}$	·1975				
$2\frac{1}{2}$	0 $36\frac{1}{2}$	·3290	0 $16\frac{1}{2}$	·3670				

TABLE 1.

The amounts of water used were 10, 20, 40, and 60 pounds per minute per inch of width of launder. Our purpose in taking the amount of water per inch of width of launder was on account of the facility it affords in calculation for the other experiments, of which the ones described in this article are only a portion.

Results.—In the accompanying results (Tables 1, 2 and 3) will be found the launder, grade, and corresponding time required to flush each material by a given amount of water. The capacity in pounds of material for each pound of water used per second has been calculated as follows:—For example, take the case of 40—60 mesh pyrite (Table 1, *a*), using 10 lb. of water per minute per inch of launder width on a $\frac{3}{8}$-in.-to-the-foot grade:

Weight of material = 10 lb.

Total amount of water = $10 \times 2\frac{1}{2}$ (width of launder) = 25 lb. per minute.

$\frac{25}{60}$ = pounds of water per second.

Resulting time = 1 hour 20 minutes 14 seconds = 4,814 seconds.

Therefore, $\frac{10 \times 60}{25 \times 4,814}$ = ·00498 lb. material transported by 1 lb. of water per second.

The results shown in Tables 1, 2 and 3 will probably be a suprise to a good many millmen. It will be seen that in nearly every case a given flow of water has a greater carrying capacity for the angular tailing and pyrite, on the low grades, in the linoleum-lined and wood launders than they do in glass-lined ones. On the other hand, the flow has a greater carrying capacity for the rounded gravel, on every grade in the glass-bottomed launder.

For example, take the case of the 40–60 mesh pyrite on ⅛ in. grade, using 10 lb. of water per inch of launder width:

The linoleum launder (Table 1, *c*), gives greatest capacity of ·01588 lb.

Next comes the wood launder with ·01118 lb. (Table 1, *a*), and last comes the glass launder with its capacity of but ·01085 lb. (Table 1, *b*).

A general inspection of the results will prove, in nearly every case, that the above statement holds true.

10 lb. 10–30 Mesh Tailings.

2½ in. Wood Launder.

Table 2*a*.

Grade	10 lb. of water.		20 lb. of water.		40 lb. of water.		60 lb. of water.	
	8·46 lb. of material. Time.	Capacity per sec. of 1 lb. of water.	Time.	Capacity per sec. of 1 lb. of water.	Time.	Capacity per sec. of 1 lb. of water.	Time.	Capacity per sec. of 1 lb. of water.
in	min sec		min sec		min sec		min sec	
⅛					29 26	·00339	13 18	·00562
¼	32 27	·0104	16 46	·0119	6 50	·0146	4 40	·0142
⅜	14 47	·0228	7 33	·0204	3 30	·0285	2 37	·0255
½	8 47¼	·0387	4 16	·0475	2 08	·0468	1 43	·0388
¾	4 18	·0787	2 7	·0945	1 09	·0970	0 56	·0715
1	2 32	·1335	1 25	·1465	0 45	·1332	0 37¾	·1058
1¼	1 43	·1975	1 01	·1970	0 31½	·1908	0 27½	·1455
1½	1 13	·2400	0 45	·2661	0 22	·2724	0 17	·2350
2	0 47	·4340	0 25	·4860	0 12	·4998	0 10	·4000
2½	0 29	·7010	0 15	·7980	0 8¼	·7272	0 7¼	·5520

2½ in. Glass Launder.

Table 2*b*.

Grade	10 lb. of water.		20 lb. of water.		40 lb. of water.		60 lb. of water.	
⅛			55 59	·00358	28 57	·00350		
¼	35 30	·0095	14 32	·0137	6 51	·0145		
⅜	16 12¼	·0207	7 13	·0276	3 16	·0306		
½	11 21½	·0298	4 22	·0465	2 01	·0495		
¾	4 05½	·0831	2 12	·0906	1 07	·0895		
1	2 30	·1360	1 32½	·1229	0 42	·1428		
1¼	1 36	·2120	1 03	·1910	0 29½	·2030		
1½	1 09	·2950	0 46	·2610	0 20	·3000		
2	0 30	·5210	0 26	·4610	0 12	·4998		
2½	0 25	·8100	0 17	·7050	0 09	·6666		

Table 2.

10 lb. Mesh Gravel.

Grade.	Wood Launder. 10 lb. of water.		Wood Launder. 20 lb. of water.		Glass Launder. 10 lb. of water.		Glass Launder. 20 lb. of water	
	Time.	Capacity per sec. of 1 lb. of water.	Time.	Capacity per sec. of 1 lb. of water.	Time.	Capacity per sec. of 1 lb. of water.	Time.	Capacity per sec. of 1 lb. of water.
in	min sec		min sec		min sec		min sec	
⅛			23 56	·00826			9 19	·0215
¼	22 03	·0186	11 13	·0178	12 58½	·0308	5 16	·0379
⅜	13 29	·0296	5 32	·0378	6 39	·0603	3 9½	·0634
½	5 53½	·0680	2 38	·0763	3 15½	·1240	1 28	·1362
1	3 34	·1120	1 40	·1200	2 09	·1855	0 53¼	·2241
1¼	2 29	·1611	1 07	·1795	1 22½	·2920	0 38	3160
1½	1 56	·2065	0 48	·2500	0 51	·4710	0 27	4440
2	1 10	·3810	0 29	4150	0 27	·8888	0 15½	7750
2½	0 47	·5030	0 20	·6000	0 17½	1·410	0 9½	·2610

Table 3.

We offer as an explanation the following: It has been noticed that on low grades the transporting action of water (on all the materials we have used) is more of a rolling and sliding one than a suspension, although the latter, of course, should also be taken into consideration.

In a glass-lined launder the tendency of the water is to run smoothly, and a sliding of the material is noticed. On the other hand, the naturally rough surface of the wood and linoleum (in comparison with glass) has a tendency to cause numerous very small eddies in the water next to the launder bottom. These eddies lift the angular particles high enough from the launder bottom (each time they tend to settle) to allow the water pressure to act on the under side of the particles and keep them rolling. It is worthy of note that the carrying capacity of 1 lb. of water decreases after the velocity of the flow in the launder has increased beyond a certain point. For instance, in Table 2 *a*, the 2½ in. launder set to a fixed grade has the following carrying capacities for tailings varying as follows with the amounts of water carried:

Group A.—

	Pounds of Water.	Carrying Capacity per Pound of Water.
Grade ⅜ in. per foot ..	10	·0228
	20	·0264
	40	·0285
	60	·0255

Group B.—

	Pounds of Water.	Carrying Capacity per Pound of Water.
Grade ½ in. per foot ...	10	·0387
	20	·0475
	40	·0468
	60	·0388

The falling off in carrying capacity at certain points is thus made plainly evident. In group 'A' the capacity of 60 lb. of water is less than either that of 40 lb. or 20 lb., while in Group 'B' the capacity of 60 lb. only just barely exceeds that of 10 lb. This having been called to the attention of the writer, he replied as follows:

'What you have noted concerning the decreasing carrying capacity of 1 lb. of water, after a certain point is reached, has been found to hold true in many cases. The only explanation I can give is that the steeper the grade becomes, the more the rolling action due to gravity must be taken into consideration.'

Now considering the rounded gravel, it is seen that it is not necessary to have eddies lift the particles in order for the water to get a rolling leverage on them. Therefore, the glass being a smoother, harder surface than the wood or linoleum, the natural result is greater capacity.

From our results it would seem safe to state that with most angular materials, the common wood launders will do as good work, if not better, on low grades, than glass-lined launders.

In plotting our curves, we found that a relation exists between grade and carrying capacity. Using the former as ordinates and the latter as abscissas plotted on logarithmic paper, the resulting curves tend to be straight lines. This fact would seem to show that some general relation exists between grade and capacity, for which formulas could be deduced."—W. C. Browning.—*Mines and Minerals*, Feb., 1909, p. 300. (A. R.)

The Dressing of Asbestos.—"The only form of asbestos which is of economic importance is chrysolite, or karystian stone. It occurs in veins of closely packed transverse fibres in serpentine rock, which, like the asbestos itself, is a hydrous silicate of magnesium. It is widely used for a variety of purposes, in most of which its non-inflammable character and its low conductivity for heat are the determining factors.

Almost the entire output, some 50,000 tons per annum, is derived from the open workings in the townships of Thetford and Black Lake in Eastern Quebec, south of the River St. Lawrence; and it is the purpose of the present article to describe in some detail the methods of dressing the mineral and preparing it for the market, which are at present employed in this district. The same form of asbestos is widely distributed, but the only localities showing any promise of deposits capable of competing with those of Canada are the Transvaal, Western Australia, Cyprus, and Newfoundland. There is a small output from Northern Italy, as well as from the United States, but it is largely of the tremolite type, which commands a lower price.

It is probable that many of the methods which are here described will prove applicable to the other deposits of chrysolite, and even, to a limited extent, to the less valuable fibrous minerals.

Classification of Asbestos Products.—The products obtained by the dressing of asbestos include crude asbestos, short fibre or 'fines,' and asbestic.

Crude asbestos consists of long fibre, usually in compact masses as it occurs in the vein. This is the material mainly employed for spinning and weaving. It is as a rule divided into two grades, No. I. in which the fibre measures over ¾ in. in length, and No. II. with fibres from $\frac{5}{16}$ in. to ¾ in. long.

'Fines' includes shorter asbestos obtained by crushing the rock. It is divided into two or three grades, the poorest in quality being described as paper stock, from its use with vegetable fibre in manufacturing paper.

Asbestic is a mixture of asbestos with finely powdered serpentine rock, which is employed in the manufacture of cement and mouldings.

In the Canadian mines the width of the veins and length of fibres vary from as much as 4 in. in exceptional cases to almost microscopical dimensions. The veins are irregularly distributed, and it has not been found feasible to raise the mineral at a profit except by open works.

A preliminary classification is made in the quarry itself. Long fibre, whether enclosed in rock fragments (which are reduced in size as far as practicable) or loose, is forwarded to the cobbing sheds for hand-dressing, while the short fibre and the rock containing it are sent to the mill for mechanical treatment.

Dressing of Long Fibre.—In the cobbing sheds the rock is broken up by sledge-hammers weighing from 6 lb. to 7 lb. The fibre is then screened in a sieve containing holes $\frac{3}{16}$ in. in diameter. The material passing through the screens and rock containing only short fibre are sent on to the mill, while the fibre caught in the sieve, and loose fibre collected in the mine, is cobbed with hammers weighing from 1½ lb. to 2 lb. on steel plates about 1 ft. square and ¾ in. thick, so that any fragments of serpentine rock adhering to the fibres are broken up and separated.

The crude asbestos thus obtained is graded and cleaned by means of sieves; those employed for No. I. grade having $\frac{9}{16}$ holes, and those for No. II. grade ⅜ holes.

In some cases the crude asbestos is separated by machinery by methods similar to those which will be described for the treatment of the rock containing the short fibre.

Dressing of Short Fibre.—Rock containing asbestos which is not long enough to be classed as 'crude' is first broken up into material sufficiently fine for the asbestos to be separated from the enclosing rock. It is next fiberised—that is to say, triturated in such a manner that the different fibres are divided one from another—and then transferred to shaking screens, whence the fibre is carried upwards by air suction, while the fine rock passes through the sieve, and any coarser material is reserved for a second fiberisation. The fibre raised from the shaking screens passes to a chamber known as a 'collector,' and thence, by way of a hopper, to a revolving screen, where it is graded into two or more qualities according to the length of fibre. Each quality then passes over a screen which allows any sand that remains to fall through, while the fibre is again lifted by air suction and carried to the 'settling chamber.' The fibre which passes through with the sand is also taken up in the same manner by air suction, and either added to the lowest grades or collected separately as paper stock.

The processes to which the rock and intermediate products are subjected, depend on the character of the material, and different arrangements are made in different mills. As far as possible, the whole is carried out automatically and continuously, the product from one operation being carried to the next by elevators or conveyors.

Drying.—At an early stage of the treatment, when it has been broken up to a convenient size, the material is dried, so that the moisture may not interfere with the subsequent processes. In favourable weather the drying may be carried out by spreading the material in a layer 2 in. or 3 in. deep on a wooden platform. It is usually, however, impracticable to dispense with artificial heat, which may be furnished by means of continuous steam piping covering the floor of a shed. One end of the piping is connected with the exhaust of a steam engine and the other with the open air, so that there is no extra expense for fuel. Sometimes a rotary drier is employed, consisting of a cylinder of boiler plate 30 ft. to 40 ft. long and 2½ ft. to 4 ft. in diameter, inclined at an angle of 7°. The cylinder rotates on friction rollers at each end, those at the lower end being grooved to keep it in position, while those at the upper end are smooth, so as to allow of expansion. It is rotated at the rate of from six to eight revolutions a minute by means of an endless chain, or gearing acting on its lower end.

The whole of the cylinder except the ends is bricked in, leaving a space of about 6 in. round it, through which pass the hot gases from a furnace at the lower end.

The material to be dried is fed in at the upper end and travels down the cylinder, which is provided in the interior with longitudinal blades. These lift the material as the cylinder rotates, and then allow it to fall through the hot air.

Crushing.—The crushing of the rock is carried out in a number of stages, the product being screened between each. The large fragments are first dealt with by a reciprocating intermittent rock-breaker of the Dodge or Blake type, fed by means of a chute, the blocks being brought forward by the attendant with the aid of a rake or pick. Sometimes two of these rock-breakers capable of dealing with fragments of different sizes are placed in series with a screen between.

After screening, the coarser material is usually dealt with in a rotary crusher, which reduces it still more. The final crushing is carried out by means of rolls, which consist of cylindrical shells of rolled steel, preferably manganese steel, fixed on cores of soft iron. The shaft of one of the rolls revolves in fixed boxes, that of the other in movable boxes, the two rolls being kept together by powerful springs, which admit of their separating should a fragment larger than usual, which might break them, present itself. In some mills automatic feeders are employed. The rolls make from two to two and a-half revolutions a second. Sometimes corrugated rolls are employed to produce a grinding action. The same result might be obtained by a differential movement between the rolls, but this has apparently not been tried.

Fiberising.—The fiberiser may consist of a horizontal shaft, provided with arms about 6 in. apart, carrying teeth or knives. It rotates in a horizontal steel cylinder lined with wood, about ten times a second. The mineral is fed in from above at either end, and the teeth or knives are so adjusted that it travels to the middle, where it is discharged through an opening in the lower part of the cylinder in a finely divided state.

Another apparatus that produces a similar result is the cyclone. This consists of two beaters of chilled iron, similar in shape to a screw propeller, placed face to face in a steel case and rotated on their axes, which are horizontal and in the same straight line, in opposite directions at the rate of about forty revolutions per second. The material treated, which is usually of the size of a walnut, is fed in on either side above the beaters, and is soon reduced to grains not exceeding a peanut in size, and fine dust which is removed by means of air suction. The beaters have to be frequently renewed, and the same is the case with portions of the lining of the case. Some of the fibre is usually torn in the process.

Screening and Suction.—The shaking screens with which this material is now treated vary from 3 ft. × 6 ft. to 6 ft. × 12 ft., and the number of pulsations is about four a second.

The pipe up which the fibre is carried by the air current is about 12 in. in diameter. At the lower end, where it takes up from the screen, it narrows in one direction to 6 in. in diameter, and in that at right angles broadens to the width of the screen. The suction is created by fans from 30 in. to 40 in. in diameter, rotating thirty times a second. The revolving screen, by means of which the fibre is graded, is provided with arms mounted on a double shaft and moving in opposite directions, for the purpose of loosening the fibre so as to effect a better separation through the meshes.

The settling chamber which receives the graded fibre is constructed with a longitudinal hopper, through which the fibre falls on to a conveyor. Both the collectors and settling chambers are provided with an inverted chimney, by which the fine dust escapes. This is sometimes collected for use in the manufacture of finishing plaster.

The conveyors and elevators employed to conduct the material from one process to another are all of the endless belt or chain type.

Where there is any chance of 'crude' asbestos being concealed in the rock, an endless rubber belt, known as a picking table, is intercalated between the rotary crusher and the rolls. Here boys are stationed, who remove any long fibre or lumps of barren rock.

Sometimes strong magnets are employed to extract pieces of iron from the rock-breakers or other apparatus, which has become mixed with the material treated, and might injure the machinery. The fine serpentine sand which passes through the screens usually contains a little fibre, and is sometimes ground down and sold as asbestic."—*Bulletin of the Imperial Institute.—London Mining Journal*, Feb. 20, 1909, p. 239. (A. R.)

TREATMENT OF THE GOLD AND SILVER PRECIPITATE AT DOS ESTRELLAS.—"At the clean-up in the Dos Estrellas mill, El Oro, Mexico, the gold and silver slime flows by gravity through launders from the zinc-boxes, first on to a 20-mesh screen, next to a 60-mesh screen, and thence to the first of two cement sumps. The short-zinc resting on the 20-mesh screen is returned to the head compartment of the zinc-boxes after being thoroughly washed: that passing 20-mesh and resting on 60-mesh is dried and melted, without acid treatment. From the first sump above mentioned the slime overflows to a second, whence it is pumped to the filter presses by a triplex plunger pump. The idea of using two sumps is to hold back the greater part of the slime in the first, thereby enabling the pump to be run at full speed till work on the boxes is finished, and the zinc room man can give more attention to the press. Then while the sumps are pumped out, short-zinc is washed and the launders cleaned. Lastly the heavy sludge in the first sump is pumped out, and the sumps thoroughly washed.

After the clean-up is completed the precipitate is partly dried by passing compressed air through the cakes in the press. The press is then discharged into a movable steam-jacketed drying car which is run under the press to receive the cakes, then returned to its place, and connected to the steam line, the cakes being left to dry all night. A small vertical boiler is used for generating steam for this purpose, and is fired for about two hours, the fire then being 'banked' and the steam allowed to drop slowly. The cakes are melted the succeeding day; they contain only about 18% moisture. Until recently the precipitate was briquetted after fluxing, but this has been discontinued on account of the amount of handling involved. At the present time the precipitate is handled only twice, as against six times in briquetting, as follows:—

Briquetting.—1. Cakes knocked from filter-frames to drying-car; and the car wheeled to place and steam connected.

2. Cakes shovelled from drying-car to fluxing-boxes; weighed; and fluxes added.

3. Mixed with shovel in boxes; and boxes wheeled to briquetting-machine.

4. Precipitate shovelled from boxes to hopper of briquetting-machine.

5. Briquettes lifted from briquetting-machine by hand and placed in boxes; and run to furnaces.

6. Briquettes placed in shovel and fed into crucibles.

Without Briquetting.—1. Cakes knocked from filter-frames to drying-car, care being taken not to break up cakes more than necessary; car wheeled to place, steam connected, dried, weighed entire, fluxes spread evenly over top of precipitate (without mixing), car run to furnace.

2. Precipitate and fluxes shovelled to crucibles.

Briquetting is a good thing, and it is also well to mix the fluxes intimately with the precipitate, but no one who has ever got some precipitate on his finger and tried to wipe it off on his overalls will dispute the statement that handling is a bad thing and spells 'loss.' Further, the precipitate at Dos Estrellas seems to melt as readily and to give as clean a slag without, as with, mixing. Care being taken not to break up the cakes as they fall from the press is almost as good as briquetting to prevent the precipitate from caking superficially and 'blowing' during fusion. The loss by dusting of such wet material is negligible. The flux generally used is :—

Precipitate	100
Borax	15
Soda bicarbonate	8
Sand	4

Scrap wrought-iron in excess.

The precipitate yields from 60% to 80% bullion. Coke is used as fuel at the two plants of the Dos Estrellas Co., but oil fuel by the El Oro and Mexico companies. Number 400 Dixon or 371A Morgan crucibles are used, each crucible holding about 86 kg. of mixed precipitate and fluxes. After fusion the upper portion of the molten mass in the crucible is poured into a conical mould with a clay-stopped tap-hole about 3 in. above the apex. The lower portion is poured into ingot mould. As soon as a shell of slag about ½ in. thick has formed in the 'tapping-mould' the clay plug is removed and the fluid central core is allowed to flow into another mould. A sample of the issuing slag is taken and granulated by being poured into water. The 'shells' from the conical moulds, and the slag from the ingot moulds are remelted, poured into conical moulds, and tapped as before. The cores are sacked and shipped at intervals to the smelter.

The slabs of bullion from the ingot moulds, together with the buttons from the tapping moulds, are remelted, no attempt being made at refining. On pouring the final bars two samples are taken ; the first from the crucible after skimming the molten metal, and the second while pouring the bars. It is of interest to note that in twelve months, during which time a total of 35·23 metric tons of bullion was melted, the total difference in value of the bullion as indicated by the two samples was only 1963·69 *pesos*. The sample taken while pouring gave the higher results.

The short-zinc passing 20-mesh and resting on 60 carries from 5% to 10% of metal and is melted with the following flux :—

Short-zinc...	100
Borax	40
Soda	20
Sand	10
Lime	5

This flux gives a very fluid slag carrying about 40% zinc and exceedingly poor in gold and silver. The resultant metal carries about 20% zinc. Being so small a quantity in proportion to the total amount of bullion, the 'zinc metal' is added to the slabs of metal which come from the screened precipitate at the second melting."—WALTER NEAL.—*Mining and Scientific Press*, Feb. 27, 1909, p. 327. (W. A. C.)

QUALITIES OF ALLOYS.—"Recent researches by Messrs. Le Chatelier, Guillet, etc., have demonstrated that the physical, chemical, and mechanical properties of certain alloys depend essentially upon the state of the different metals entering into their composition. Thus, the micrographic studies of copper alloys made by Mr. Guillet demonstrate existence of seven different solid solutions, according to the percentage of copper, presenting particular crystallisations.

Manufacture of alloys always comprises three phases—melting, casting the re-melted metal, and commercial preparation of the alloy. Usually the raw materials are waste products—shreds from plates, filings, etc. In the case of zinc, there is some loss to be estimated in melting. When the metal has to be subsequently rolled or worked, sand, clay, or chill-moulds and ingots are employed. Rolling should be executed hot, with brass containing 55% to 60% copper ; if the percentage of copper is greater, the metal should be rolled cold. The most usually employed special brasses are lead, tin, manganese, and aluminium brass alloys. Lead is in small percentages of 0·5% to 3%, placed in the crucible a few moments before casting.

Tin gives a special resistance to sea-water, and, consequently, is much employed for brass for ships. The percentage of tin in the brass must not exceed 2% for the alloy to be forgeable. High resisting bronzes are more and more employed. Greater attention is now rightly given to their mechanical properties—tensile strength, elasticity, elongation after rupture, etc.—than to their centesimal composition. Great care must be given in melting the alloy and in the construction of the moulds, which should allow the gases produced in casting to easily escape. Bronzes should be manufactured at the lowest possible temperature, then re-melted at a sufficiently great temperature to obtain a good run and metal free from blisters. To estimate the temperature, an iron rod is dipped into the liquid copper. If the metal adheres to the rod, the alloy is not made, and if the rod gives off fumes the metal is too hot. An intermediary temperature between these two points must be selected."—*La Fonderie Moderne.*—*London Mining Journal*, Feb. 20, 1909, p. 263. (A. R.)

MINING.

MINE LABOUR-CHART.—"Stripped of its romantic possibilities, mining is a commercial business, carried on for the profit there is in it, and the business of the manager, in its ultimate analysis, is to make the profit as large as possible.

The general manager of a mining company should keep himself personally familiar with all the details of its operations. If he fails to do so, chaos results, labour becomes inefficient, costs go up, and profits go down. Nowhere is this emphasised more than in Mexico.

If the mining operations are extensive and complex, involving, besides the work of mining the ore, its transportation and treatment by various milling operations, and the marketing of the products, the business, to be carried on properly, must be divided into a number of main departments, which are sub-divided into minor departments. Each main department should be under the immediate charge of a superintendent, or a foreman, and each minor department under a shift-boss. Organised in this way, and with all the principal centres of activity connected by telephone, the manager can keep in touch at all times with the details of every branch of the business, and be in a position to perform effectively the full duty of manager.

The total cost of mining operations may be broadly divided under the heads of labour, supplies, and marketing. Of these, it will be found, on averaging a number of cases, that the cost of labour will be about two-thirds of the total cost. The importance, therefore, of an efficient labour department will be apparent. This department must be sub-divided and organised differently, according to the peculiar conditions at each mine, and no one system, however well suited for one particular concern, will exactly fit another. But any standard system may be modified to suit any particular case.

Each foreman of a given department should be furnished with suitable blank forms, on which he

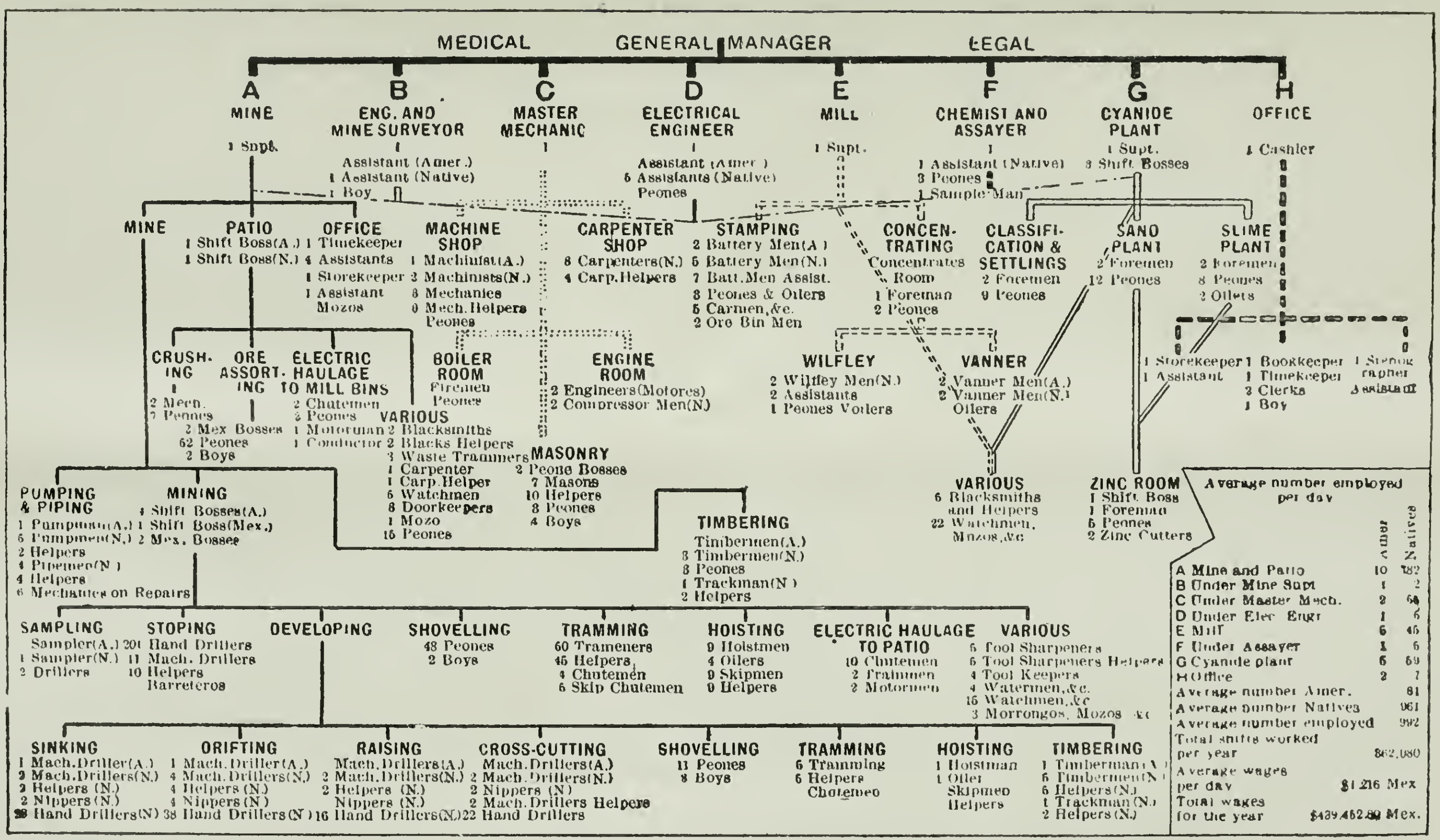


Fig. 1.—Guanajuato Consolidated Mining & Milling Co. Staff and Distribution of Labor for 1907.

should record, at the end of every shift, the main details of the operation of his department. These blanks, when filled out, pass through the superintendent's hands, and, with his remarks and explanations, are presented to the manager for examination and filing. Thus, within 24 hours, the record of the work done in every departmental sub-division of mine, or mill, together with its cost, is before the manager for study and comparison with previous records.

It is not always possible, in fact it is very difficult, to get foremen, or shift-bosses, who are in every way competent to supervise the departments, but the best men may be selected for this purpose, who, if properly supervised and encouraged by the manager in the work, will give profitable service.

The work of the mine manager will be facilitated by devising a labour-chart which will show diagrammatically the various departments of labour, the number and class of men employed, the cost, and the method of supervising each class.

The foreman of each department should have a labour-chart pertaining to his own special department, and all charts of the various superintendents, combined on one sheet, should make up the manager's chart, which will serve also as a 'flow-sheet' of authority in the organisation. A labour-chart may show the distribution of labour for a month, or a year, or any other period.

The labour-chart I used for the Guanajuato Consolidated Mining and Milling Co. was included in my annual report to the directors covering the operations of that company during 1907, and with the permission of the President of the company, F. G. Corning, I submit this chart, Fig. 1, for the purpose of illustration. It shows the labour classification and the cost for the operations of the company during the year. The chart, in its original form, was printed in colours, so that the eye could readily distinguish the ramifications of the various sub-divisions. The general division was in black and blue; mining, in red; milling, in purple; cyaniding, in light-brown; office work, in dark-brown. In Fig. 1, each colour has been replaced by a distinctive line. The method of arranging the details of labour classification might be more advantageously shown than is done in the chart, but it is hoped that it will serve as a suggestion for a system that renders more efficient the management of mining operations on a large scale."—JOSEPH MACDONALD.—*Bulletin of the American Institute of Mining Engineers*, Jan., 1909, pp. 77-80. (A. R.)

UNDERGROUND MINING.—"For very deep mines it seems to be coming now to be much the practice to sink circular shafts, and I find that throughout the world the dimensions of these vary very greatly. In India, at the Mysore mine, I have to do with the sinking of a vertical shaft, which has now reached the depth of 2,628 ft., which has been sunk of a diameter of 21 ft. outside the brickwork, or 18 ft. inside. On the Witwatersrand, in the two or three cases where circular shafts were adopted, I believe the diameter varies from 15 ft. to 18 ft. At the Dolcoath mine in Cornwall a circular shaft 18 ft. in diameter is being sunk to a projected depth, I understand, of 500 fathoms. At Clausthal I know of a shaft projected to 2,900 ft. of depth, of a diameter of $15\frac{1}{2}$ ft. I should, perhaps, have alluded to rectangular shafts first, as probably 99 out of 100 of the metalliferous mine shafts in the world are of that shape, and they, of course, vary in size and design to accord with the purpose for which they are sunk. I understand that the Lake Superior shafts, through which enormous quantities of cupriferous material have to be hauled from great depths, have usually been rectangular; for instance, the Tamarack mine shaft being 29 ft. 2 in. × 8 ft. 10 in., the Anaconda 20 ft. 4 in. × 6 ft. 8 in., Red Jacket shaft, which is some 4,900 ft. deep, is a four-compartment shaft 25 ft. long, but whether these measurements are within or without timbers, I am not sure, but probably within. On the Rand, shafts are already in existence 42 ft. (?) long, including six hauling ways and one pump and ladder way, and I believe it is proposed with very deep shafts to increase the length to over 40 ft. (?) I have this morning been informed that the Catlin shaft on the Jupiter mine is sunk 4,077 ft. vertical, and 673 ft. further on an incline of $40\frac{1}{2}$°; reducing the latter figure to vertical measurement, a total depth of 4,515 ft. vertically from surface has been reached, and this, I believe, is the lowest reef working on the Rand. I am further told that there are several other shafts which have struck the reef in the vicinity of 4,000 ft. depth. In my experience it has always been the custom to sink rectangular shafts in the earlier stages of progressive mines, and where operations are commenced in out-of-the-way places, with but few skilled hands, this is almost invariably the case.

In regard to the drivage of levels, stoping and underground tramming, I find that the practice varies enormously over the world, obviously, I take it, being governed by reasons of hardness of rock, width of reef or lode, quantities of rock or ore to be removed, and distance to be traversed. In level driving or tunnelling we have three classes of undertakings all intimately connected. First, great civil engineering works like the Simplon Tunnel; secondly, big mining adits like the Sutro Tunnel, on the Comstock lode, in Nevada, and the Newhouse Tunnel, in Colorado; and thirdly, the more usual sizes of mining drifts or crosscuts. A work like the Simplon Tunnel cannot fail to be of interest to the miner, although in many ways carried out under conditions somewhat different to those of ordinary mining work. This tunnel, from Brigue, in Switzerland, to Isella, near the old Crodo Gold Mines, in Italy, is 19,770 metres long, or a little over twelve miles. It is noteworthy that this tunnel, like so many others of a similar kind, failed absolutely to discover any mineral wealth. The depth below the peak of the mountain range being 7,000 ft., this may be said to be the deepest mining work in the world. In regard to speeds of driving mine levels, I have before me a statement which, if substantiated, might claim to be a record. Messrs. R. Meyer, of Mulheim, claim to have advanced 465 ft. in one month's drivage in trachyte in 1905, while their average speed for several months in limestone is claimed to have been 294 ft. per month. I understand that the 3,100 ft. level of the Chollar Potosi mine on the Comstock lode was driven many weeks at the rate of 80 ft. per week, or 340 ft. per month, and that at the Newhouse Tunnel, Idaho Springs, Colorado, driven north into Gilpin County, a speed of 300 ft. per month was attained, and about 3,000 ft. driven in one year in hard granitic-gneiss and schist. I believe that even in the hard formation of the South African mines, a speed approaching 300 ft. per month has been attained. Stoping is a subject which has received a vast amount of attention by mining engineers of recent years. It is a subject so wide and so much dependent upon local conditions that I will not attempt to deal with it at any length. It appears to me that it is almost invariably a question for the man on the spot, who has to evolve for himself the best method of taking down the ore in his mine. Take, for instance, the Broken Hill mines, where they have ore bodies, say, from 15 ft. to 300 ft. in width, averaging, perhaps, over 100 ft. Here a great cost has to be incurred in timbering and filling, the hanging walls being very insecure. On the other hand, at the Alaska Treadwell, in quite another part of the world, where the ore bodies have even greater width, up to 420 ft. no timbering is, I believe, needed, at any rate as long as the stopes are kept two-thirds full. Personally, I have had little experience in mechanical haulage in mine levels and stopes, but I understand that shaking conveyors in stopes, mechanically operated continuous ropeways on tramroads, and electric locomotives, where the length of level is very great, are being largely adopted.

A few words on the actual breaking of ground. Here, too, progress has been continuous and rapid, the greatest advance being the substitution of high explosives for the now almost obsolete gunpowder in blasting. In fact, it would not be too much to say that the invention of high explosives and the safety fuse mark the two most forward steps in the mining history of the last century, and to these, more than to any other cause, is to be attributed the rapid speed of development possible to-day. High explosives, in contradistinction to black powder and the like, have been defined as explosives which require initial detonation—*i.e.*, the use of a detonator. It was Mr. Alfred Nobel who, prior to his invention of dynamite, conceived the idea of this initial detonation, which he first applied to nitro-glycerine as early as 1864. Dynamite, which was invented in 1867, was the first high explosive used, and remained practically the only explosive of its kind until Nobel, in 1875, invented blasting gelatine. The figures given in a paper read by the inventor before the Society of Arts, in May, 1875, show the quantities used in the first years of the introduction of dynamite to have increased from 11 tons in 1867 to over 3,000 tons in 1874 With the introduction of blasting gelatine in 1875, and the modifications known as gelatine dynamite and gelignite, the use of nitro-glycerine explosives became more and more general whenever hard rock had to be dealt with. In 1864, explosives of the 'Sprengel' type, the first representative of which was a nitrate of ammonia compound known as the 'Favier' explosive, termed in this country 'Miners' Safety Explosive,' were introduced, and in course of time these nitrate of ammonia explosives, of which numerous modifications were devised, have become the leading explosives in coal-getting, because of their relative safety in the presence of firedamp. Several nitro-glycerine explosives, such as carbonite and saxonite, in which the temperature of the flame is reduced either by an excess of carbonaceous substance or by the addition of salts, such as oxalate of ammonia, sulphate of magnesia, alum, etc., have likewise been added to the list of so-called safety explosives. The displacement of black powder and other similar mechanical mixtures by high explosives was concurrent with the recognition of the relative cost of the explosive plus detonator and fuse as compared with the cost of the borehole, the harder the rock the higher is the cost of the borehole and the more desirable it is to blast with high explosives of high density, thus obtaining the greatest amount of cubage with the least relative cost. Thus nitro-glycerine explosives held their own and increased their field, whilst nitrate of ammonia explosives and those which, like them, are of lower density, were confined to coal getting and workings in soft rock and stone where the borehole was not costly. The relative densities are, roughly, 1·6 for nitro-glycerine explosives and 1·1 for nitrate of ammonia and chlorate mixtures, the density of tonite, which is a guncotton nitrate of barium mixture, being about 1·2. It is self-evident that the advantage of density being recognised, black powder was bound to be replaced by high explosives wherever the borehole was difficult to make. The introduction and gradual development of rock drilling machines still further accentuated the necessity of using explosives of high density and considerable strength, as it became possible to make the holes deeper and relatively less in number. Thus, for instance, mining on the Rand is almost exclusively done with blasting gelatine, which is the strongest explosive known, whilst in quarrying less powerful explosives are needed. The evolution in the use of high explosives is therefore essentially an economic question, although, on the other hand, it is of importance to remember that certain classes of rock can only be dealt with by means of high explosives, the selection of the right explosive for the particular work being therefore of considerable moment, and the relative cost of boring has always to be borne in mind."—EDGAR TAYLOR, Presidential Address, Inst. M.M. — *London Mining Journal*, March 20, 1909, p. 369. (A. R.)

THE DEEPEST BOREHOLE IN THE WORLD.—"Hitherto the deepest borehole in the world has been that sunk near Rybnik (Upper Silesia), namely, 2,003 metres; but this depth has now been exceeded by a boring at Czuchow, in the same neighbourbood, which has already reached 2,156 metres, and is to be continued to 2,250 metres, if possible. The hole was commenced in December, 1906, with a diameter of 16 in., but in the following May the bits were replaced by diamond crowns, the diameter being reduced to 7 in."—*Montanistische Rundschau.—London Mining Journal,* March 13, 1909, p. 343. (A. R.)

PETROL LOCOMOTIVES IN MINES.—"One of the recent digressions in Westphalian mining circles has been the introduction of petrol or benzine locomotives for underground traction. The idea of the introduction of such volatile and highly-inflammable liquids as petrol or benzine into coal mines would strike the uninitiated as appalling; but, be that as it may, the difficulties and dangers have been sufficiently overcome, and such locomotives are actually at work in the Westphalian coalfields. In the mines of a single association in Dortmund 16,562 tons out of a total annual output of 94,145 tons—that is, above 17·5% of the whole production—have been drawn by petrol locomotives underground. Of the rest of the output, 63·5% was drawn by endless rope haulage, 15·75% by electric locomotives on the trolley system, smaller quantities were dealt with by electric locomotives with accumulators or by endless chain, and the least amounting to about ½%, by the main and tail system of haulage.

The locomotives used for the underground traction with petrol do not differ essentially from those in ordinary use, the chief characteristics in connection with them being the nice arrangement of the different parts to accommodate from 8 h.p. to 16 h.p. in a very restricted area, and the precautionary devices introduced to provide for security against the risk of explosion in handling and in the use of these highly inflammable and volatile liquids, and in the disposal of the products of their combustion. In the early stages the charging of the petrol locomotive was effected by means of oil-tight tanks fitted with adequately protected orifices, which were filled above ground and sent below to be attached to the locomotives, but the leakage of the connections proved a prohibitive obstacle to this arrangement. Sending the locomotive to the surface in the cage to be filled was next contemplated, but as it would have necessitated even a still more cramped contruction in the engine, that idea was abandoned, and therefore attention was directed to making special arrangements for charging below ground."—*The Times.—Transvaal Leader*, March 19, 1909. (A. R.)

MISCELLANEOUS.

Geochemistry.—"Geochemistry is the coming form that geological science is taking, and is the most up-to-date science. Eighty chemical elements are known. Most of them are found only in combination. These compounds are minerals. Aggregation of minerals form rocks. A rock represents a chemical system in which, by various agencies, chemical changes are brought about. The study of these changes is the province of geochemistry.

Reaction takes place between the constituents of the crude ore: reactions due to its aqueous envelope, reactions in volcanic magmas and during their contact with rock masses, are studied, changes due to percolating waters, superficial action of the atmosphere. The solid crust of the earth is made up of three shells or layers. The innermost of crystalline plutonic rocks approaching the original material of which the crust was formed. Next, the overlaying layer of sedimentary rocks, and above this soils and unconsolidated material.

The elements were evolved from one primal stuff. The surface layer of the earth to a depth of 10 miles consists of 93% solid and 7% liquid matter. The volume of saline matter in the ocean is 4,800,000 cubic miles, or enough to cover the entire surface of the United States 1·6 miles deep. Hence, it is not surprising that the beds of rock salt at Stassfort are so great and deep. The atmosphere consists of nitrogen and oxygen.

Whenever animals breathe or fire burns, oxygen is being withdrawn from the air and locked up in compounds. By growing plants under sunlight, carbon dioxide is decomposed and oxygen is liberated. The atmosphere is being slowly depleted of its oxygen.

Cotopaxi alone emits more carbon dioxide annually than is generated by life and combustion in a city like Paris, which throws 3,000,000 cubic meters of the gas into the air. Rain and snow collect chemical impurities from the atmosphere and redistribute them upon the soil. Chlorine is abundant in the air only near the sea, and diminishes as we recede from the coast. An atmosphere of volcanic origin must sooner or later be exhausted, as in the case of the moon.

In the ocean, some 18 to 20 substances have been detected, including nickel, copper, lead, silver, gold.

With regard to gold, its presence in sea water was shown by E. Constadt, in 1872; such as 5 mgs. gold and 20 mgs. of silver per ton. Gold is present in the ocean in thousands of millions of tons. In the Bay of San Francisco, L. Wagner found, per metric ton, 11·1 mgs. of gold and 169·5 mgs. silver. Plants and animals are almost incredibly abundant in the ocean. These absorb carbon, hydrogen, nitrogen and oxygen to form the principal constituents of their soft tissues; carbon dioxide being abundant from their life, it dissolves much calcium carbonate, to form their hard shells.

Decomposing organic matter reduces sulphates of sea water to sulphides, which, by reaction with carbonic acid, yield sulphuretted hydrogen. Some hydrogen sulphide reacts upon the iron silicates of the sea floor, forming pyrite. The circulation and distribution of carbon in the ocean is as much influenced by living beings as by exchanges between the sea and atmosphere.

The waters of closed basins akin in size to inland seas are those in isolated areas from which the drainage never reaches the sea. Streams originate in the higher portions of such areas; their waters gather in depressions, and ultimately, by the concentration of their saline constituents, form salt or alkaline lakes or dry beds of solid residues. The latter condition is in very arid areas, where evaporation is so rapid that no large body of water accumulates. The important closed basins are permanent reservoirs, such as the Caspian, Great Salt Lake and the Dead Sea of modern times, and Lake Bonneville and Lahontan of ancient date. The Great Salt Lake is the diminished reminder of the Great Lake Bonneville.

The salinity of Great Salt Lake far surpasses that of the ocean. All tributary waters contain quantities of carbonates absent from the lake itself; the strong brine of the lake not being able to hold calcium carbonate in solution. The waters of the Dead Sea are bitter, low in sodium, high in magnesia, rich in bromine.

Alkaline waters are representative of volcanic regions: saline of sedimentary deposits.

The least soluble salt, such as gypsum, will form first, the most soluble, like rock salt, will remain longest in solution.

In mineral springs, one water may rise from beds of salt, another from solutions of gypsum, a third from pyrite. Hot springs often deposit silica in the Yellowstone Park. Alkaline waters, especially if hot, are solvents of mineral substances; hence the quartz of veins as well as the ores.

During a volcanic eruption, the gaseous emanations are the first to appear, and after the eruption is over, gases are given off from the cooling lavas, as well as from hot springs and solfataras. Steam is the commonest gas. Other gases are hydrogen, oxygen, carbon-monoxide, hydrochloric acid, chlorine, methane, hydrofluoric acid, silica fluorides. Deposits from these are sublimates, the commonest being native sulphur. Silver was found at Cotopaxi, ejected as a vapour; also ammonia. Volcanic gases produce volcanic explosions; deep-seated magmas are impregnated with gases which explode on relief of pressure.

Volcanic gases appear in a certain regular order.

1. Gases issuing from an active crater, owing to excessive heat, are dry.

2. Hydrogen burns to form more water vapour.
Carbon gases oxidise to carbon dioxide.
From sulphur, sulphur-dioxide is formed.
Steam reacting on metallic chlorides generates hydrochloric acid and acid fumaroles appear.

3. Gases force their way through cracks in lava, corroding the rocks, and where sulphides occur, hydrogen sulphide is set free. If carbonate rocks are met, carbon dioxide is liberated.

4. Only steam with carbon dioxide remains, and the latter soon disappears.

We can measure the temperature at which lavas fuse under pressure.

Pressure steadily increasing with depth, the melting point must be raised, but water in the molten mass tends to lower it. Pressure also prevents the escape of dissolved vapours and increases fluidity.

Temperature increases with depth.

Lava flowing from a crater is heavily charged with steam. When the pressure is relieved, gases escape and explode. The magma previous to eruption, is a mass of rock-forming matter, in a state of fusion, heavily charged with gases under enormous pressure. Different lavas issue from the same opening during a series of eruptions, and within a given area there are a variety of igneous rocks related to one another and showing a common origin. From these differentiation of minerals take place."—F. W. Clarke, Bulletin No. 330, U.S. Geological Survey.—*Mining Science*, March 11, 1909, p. 190. (A. R.)

The Properties of Matter—Solid State.—"Prof. Thomson gave a brief *résumé* of his remarks on the subject of gravitation, and said that from the practical point of view it was a great pity that we could not alter the weights of chemical masses. If, for instance, a body changed its weight as it passed from the solid into the liquid state a very good engine could be constructed by alternating the states and balancing one against the other. This would be very nearly, but not quite, perpetual motion. But the most careful research had failed to show that any change of weight took place under any known conditions, no change having been detected that could be due either to chemical combination, temperature, crystallisation, or to the nature of a medium. Prof. Landolt, indeed, had made some experiments which seemed to indicate that a slight change of weight took place on chemical combination, but the German physicist's latest conclusion was that the apparent change was due to a slight alteration in the bulk of his glass vessels produced by heat. Other experimenters sought for possible changes of weight due to the action of heat. Did a body weigh exactly the same at different temperatures? The experiments made by Prof. J. H. Poynting and others certainly showed that there was no change of weight, to within one part in one million, produced by the alteration of 1° C. of temperature. Prof. Poynting had also investigated the possible effect on weight of crystallisation by testing bodies placed at right angles to each other. Another question referred to the possible effect upon weight of the nature of the medium. Did two bodies, for example, attract each other with the same force when separated by air, or by iron, or by water? It had been shown by Prof. Boys, using a modification of the Cavendish apparatus, as described in a previous lecture, that no perceptible change of weight could be attributed to the influence of any medium. Gravity in fact had no weak parts, and could not be affected by any change of circumstances known at present. This made it difficult to formulate any adequate explanation of gravity.

The one hopeful feature about gravity was in the consideration of the proportion between the weight and mass of different bodies. There were various methods of obtaining the value of mass by weighing. After describing Kater's pendulum, Prof. Thomson exhibited the bi-filar pendulum invented by Prof. Horace Darwin. This device depended upon the fact that the opposite sides of parallelograms are equal. In Prof. Darwin's compound pendulum, measurements are made on a table swinging from two wires or tapes so arranged that the table is invariably horizontal. Some of the most important investigations of weight were made by Bessel, who proved that the ratio of weight to mass is the same to within one part in 60,000. Since Bessel's time radio-active bodies had been discovered and there was now some reason to believe that the ratio of weight to mass for such bodies was different from that of non-radio-active bodies. He would attempt to explain this possibility by supposing that a sphere were swinging in water. The sphere, as it moved, would displace some of the water, and if it were sought to determine the mass of the sphere, a part of its mass would be due to the mass of the water. In the same way there was reason to believe that in the case of a radio-active body moving through ether there is a like displacement; and in consequence of the ether having mass, a part of the mass of such a moving body would be due to the mass of the ether. The proportion of the mass due to the ether would be one part in 650,000. Prof. Thomson said that he had made many experiments in this direction and had used a radium pendulum. Although he had only a few milligrammes of radium for the experiment, yet he had found that there was no change observable to within one part in 2,000. They had recently set up another pendulum at Cambridge to conduct a similar test with uranium, and had already shown that there was no change in the proportion of mass to weight of uranium to one part in 100,000.

There was no lack of theories as to the nature of gravity. In preparing his lecture that morning Prof. Thomson said that he had looked up more than 25 important theories of gravitation, any one of which might be true if there were any means of testing it. There were two theories which attracted considerable attention. One theory supposed that throughout the universe there were particles moving with great speed in all directions and that the earth, so to speak, was almost but not entirely permeable by these particles. The sun, acting as a kind of umbrella, shielded the earth from the bombardment of these particles on one side, and the earth was thus supposed to be pushed towards the sun. The chief objection to this theory, as advanced by Clerk Maxwell was that it involved the assumption that for each gramme of matter there was a transformation of energy equivalent to more than 1,000 h.p. The second theory of gravitation worth mentioning was that touched upon in the previous lecture. This supposed that in the case of electrified bodies the attraction between unlike particles is always a little more that the repulsion between like particles. There was now some hope of testing this theory, which was at least as old as Franklin. They had just succeeded at Cambridge in obtaining positively electrified particles which had apparently only half the weight of the hydrogen atom. It was believed that the quantity of electricity contained by an electrified particle was proportional to its atomic weight, and, assuming that the charge on a corpuscle could be taken as a unit, there was just a possibility of discovering if any change in the relation of mass to weight occurred; but the mathematical and other difficulties which the process would involve seemed hardly likely to justify the time that would be required for the investigation.

Turning to the discussion of the existence of matter in its solid, liquid, or gaseous state, Prof. Thomson said that the broad lines of distinction were very marked, yet the transition from one state to another proceeded by steps so small as to be hardly distinguishable. He showed a specimen of hard pitch flowing out of a funnel which had been kept in a vertical position for 22 years, and said that sealing wax if gently treated could almost be twisted into a knot. An enlarged micro-photograph of cadmium showed that this metal has a crystalline structure something like that of a wall built of loose stones. When such a metal was strained a good deal of energy was expended. In this regard different metals behaved in different ways. The phenomenon of 'elastic fatigue' was then illustrated by means of vibrating wires. Some metals, the lecturer said, were 'born tired,' and the leaden wire exhibited was merely 'pretending to vibrate.' Vibrating wires resembled human beings; after vibrating all the week, they got tired on Saturday night, but if allowed to rest on Sunday, they were refreshed by the holiday. It had lately been found out that wires were also refreshed if subjected to warm baths. Illustrating the changes which metals undergo at particular temperatures, Prof. Thomson showed the phenomenon of recalescence by means of a plate of red hot iron. The effect of changes of temperature on iron was also shown by means of an electrically heated wire, the variation in the length of which was shown by its shadow moving across the screen. He also demonstrated how iron lost its magnetic quality at a high temperature and regained it on cooling. By means of a rotating weight suspended from a vertical spring, the lecturer showed in conclusion how the vertical movement of the spring could be transformed into the rotating movement of the weight and *vice versa*."—Prof. Sir J. J. THOMSON, Lecture Royal Institution. — *Times Engineering Supplement*, March 24, 1909. (J. A. W.)

Reviews and New Books.

(We shall be pleased to review any Scientific or Technical Work sent to us for that purpose.)

A MANUAL OF PRACTICAL ASSAYING. By H. VAN F. FURMAN, E.M. Sixth Edition. Revised by William D. Pardoe, A.M. $3. (New York, U.S.A.: John Wiley & Sons.)

"A new edition of this standard work will be welcomed by all, and the book has not suffered in the hands of the reviser who undertook the revision on the death of Mr. van Furman. Much new matter has been added, and some modifications have been made, and the work continues to be one which should hold its own among the many of its class which have recently appeared.

As the reviser points out in his preface, the chapter on zinc has been re-written, while chapters have been added on special metals, etc. The wisdom of entirely replacing the old volumetric ferrocyanide method by a comparatively new modification may be doubted, even though the new one may be an improvement. Many prefer the old method, and have now to refer to an older edition or to another author to find it described. In the case of the so-called rare metals, their growing importance deserves somewhat fuller treatment than they at present receive. The method of assay given for tungsten would, for instance, be scarcely applicable to tailings from wolfram ores.

It is curious also that, although methods other than those published in American journals receive some notice, many very important ones are totally ignored. The section on tin is totally inadequate, and entirely ignores the only methods which are of any use for low grade ores. The methods of Pearce and Beringer are not even mentioned. Similarly, in the case of antimony, the 'bromate' volumetric method is ignored, and throughout the work there is a tendency for the author to act as final arbiter as to the best method. Such an attitude is scarcely to be recommended, although both author and reviser are qualified to advise.

Perhaps a word as to the index may result in some improvement in the next issue. A search for a description of the assay of telluride gold ores, on which there is a most admirable chapter, was unavailing under the letters G and T, but it was ultimately unearthed by turning up D, where the ingenious indexer had buried it with many other unfortunates under Determination of—.

Special praise should be accorded to the thoroughly practical tables, and, above all, to the sections relating to such matters as the calculation of blast furnace charges, the mechanical testing of gold and other ores, and the many other practical matters for which alone this work would deserve to occupy a prominent position on our shelves." — *The Mining Journal* (London) *Literary Supplement*, March 27, 1909. (W. A. C.)

GOLD: ITS GEOLOGICAL OCCURRENCE AND GEOGRAPHICAL DISTRIBUTION. By J. MALCOLM MACLAREN, D.Sc., F.G.S., F.R.G.S., M.I.M.M.; late Mining Specialist, Government of India; formerly Assistant Government Geologist, Queensland, etc. Royal 8vo. 662 pp. With one coloured plate, 37 plates, and 213 illustrations in the text. 25s. (London: *The Mining Journal.*)

"The basis of any genetic classification of necessity takes one back to those considerations of the earth's crust and interior which bear upon ore deposition, and with this the author starts, giving a statement of the views which represent the concensus of present opinion on this subject. He then proceeds to give in some detail the occurrence of gold in Nature; to describe its forms as native metal, and its associations as compound; and then, on p. 42, the classification of auriferous deposits is begun.

After refusing the classifications based on form, matrix, or associated minerals, as being of obviously adventitious character, it is stated that the most natural grouping of the world's gold deposits appears to be reached by a combination of geographical and geological data, resulting in the establishment of fairly definite auriferous provinces, well separated from each other in time, in space, or in both. A tabular classification is given on p. 44, and afterwards a general description of the various groups.

An examination of the table on p. 44, and a reading of the general descriptions following, shows the grouping of the world's gold deposits to have been into two main divisions, primary and secondary—the former including those of which no prior state is known, whereas the latter are those obviously or presumably derived from the previous primary deposits. The interest of the book is mainly in the primary occurrences, and a principal division of these is made into (a) those connected with andesites or diabases, and (b) those with acid rocks of a granodioritic type. Each of these divisions provides sub-divisions, according to the geological periods when deposition is held to have taken place; it is under these last sub-divisions that the general descriptions are given. A further general description of the secondary deposits then follows, in which, naturally, alluvial gold figures largely. It is rather remarkable that it is under this description of ore deposits that the genesis of the Witwatersrand goldfields is discussed, though the contemporaneous deposition of gold and pebble is refused.

The source and transport of the gold next occupy attention, and then its deposition and concentration, extending to p. 118, where the first part of the work ends.

The second part, which extends to the end, at p. 662, deals with the geographical distribution of the occurrences, the description being under the headings of continents, countries, and districts. This part is a valuable collection of much information concerning the many goldfields. There are full descriptions of most of them, and the geological circumstances of each are so set out, that this book should be of great help to those who are subscribers to the various mining periodicals. It forms a basis whereby one occurrence may be connected with another in a far distant country, and a new interest is thus given to the descriptions of regions which the reader may not himself have seen.

With regard to the Witwatersrand goldfields, an inadequate description is given. There are no sections in detail of the reefs, nor are any facts stated in support of the importance attached to the influence of the diabase and dolerite dykes. In giving the placer theory of the genesis of the gold, it is not stated that it is the 'marine' placer theory which is advocated, though the fact of it being marine is one of the essentials of that theory. It is moreover hardly correct to say that Gregory, in order to account for the extreme fineness of the gold, assumed the original placer gold to have been taken up in solution and redeposited. This was so assumed to account for the crystalline character now found ; the marine placer theory in itself accounts for the fineness. It is also remarkable that no mention is made of the fact that the auriferous conglomerates of West Africa are very similar in all essentials to those of the Transvaal, though other more differing conglomerates are thus compared.

The print, paper, and style of the book are very good, and the book makes a handsome volume ; the indexes also are full and complete. With regard to the illustrations ; it may be said that the photographs are very good, but that the reduced plans and maps are less satisfactory, notably the one given on p. 46, supposed to assist in the definition of the auriferous provinces. Others suffer from confusion of detail, brought about by reduction of size. This detail would, in the writer's opinion, in most cases have been better left out, or the geological conditions would have been better conveyed, if they had been rendered somewhat diagrammatic.

The work covers so wide, and in some respects, controversial a field, that a review almost necessarily takes the line of criticism ; perhaps in a future edition the author will have something to say on the points raised."—*London Mining Journal*, April 17, 1909, p. 495. (A. R.)

THE RHODESIAN MINER'S HANDBOOK. By F. P. MENNELL, F.G.S. Second Edition. (Bulawayo, Rhodesia : Ellis Allen.)

"This book contains information valuable to Rhodesians and of interest to others connected with that country. The subjects dealt with are necessarily of a general character.

A description of the diamond deposits appears, and should be read. So far the industry has not passed much beyond the prospecting stage, and data as to the cost of winning the stones do not appear.

The geology of the country, a subject Mr. Mennell is well able to discuss, is described ; but the absence of fossiliferous remains, except those found in the coal measures, makes the determination of the ages of most rocks in that country a matter of difficulty. Gold apparently furnishes over 93% of the total value of the mineral output, including coal, and for some time is likely to continue of similar proportions.

One reason, we think, why base metals are not more generally worked, a point Mr. Mennell does not seem to have mentioned, is the high railway rates existing.

The uses of the sieve in prospecting, are well emphasised in the book. The volume generally, is made up in an attractive form, and contains some good photographs of the country."—*London Mining Journal*, April 24, 1909, p. 536. (A. R.)

MINING METHODS IN EUROPE. By LUCIUS W. MAYER. 8vo., pp. 169, ill., index. (New York : Hill Publishing Co., 1909.) Price $2.50 net.

"The elaboration of the notes taken by a technical man on his rambles, constitutes a contribution which is often more striking than a formal treatise. No attempt at systematic presentation is made. Methods are described as they come to the traveller's attention. A large part of the work is devoted to coal, iron, and slate mining in Great Britain. Cave-mining is represented, but details are wanting. It would be impossible for a mine superintendent to adapt much from Mr. Mayer's account of the caving system as practised in its early home. In the account of longwall work, however, a mass of detail and lucid description, aided by diagrams, enables the reader to appreciate the processes in principle and application. This part of the work, and the minute accounts of the flushing system of mine-filling as practised in Germany, give to the book a high importance. In these particulars it may be regarded as a most valuable presentation of data but little known in America. There are many typographic errors, and others that should have been eliminated in editing. For example, an entire chapter devoted to the famous Cleator Moor mine in Cumberland, is gravely dedicated to Creator Moor."—*Mining and Scientific Press*,, April 3, 1909, p. 495. (W. A. C.)

Selected Transvaal Patent Applications.

RELATING TO CHEMISTRY, METALLURGY AND MINING.

Compiled by C. H. M. KISCH, F.M.Chart.Inst.P.A. (London), Johannesburg (Member).

(N.B.—In this list (P) *means provisional specification, and* (C) *complete specification. The number given is that of the specification, the name that of the applicant, and the date that of filing.)*

(P.) 171/09. Wilhelm Mauss. Improvements in automatic percussive apparatus. 8.4.09.

(P.) 172/09. James Weatherston (1), John Alexander Lawson (2). Improvements relating to electric lamp holders. 8.4.09.

(C.) 173/09. Robert Watson Scott. Improvements in methods of smelting. 10.4.09.

(P.) 174/09. Edmund Jerrard. Improvements in and relating to clearing, removing or transporting broken ground in or from underground workings of metal and diamond mines and in apparatuses therefor. 10.4.09.

(C.) 175/09. Heinrich Wohlwill. Process for refining gold. 10.4.09.

(C.) 176/09. Heinrich Flottmann. Improvements in and relating to water feed devices for rock drills and the like. 10.4.09.

(C.) 177/09. Heinrich Flottmann. Improvements in and relating to water feed devices for rock drills and the like. 10.4.09.

(P.) 178/09. Henry Samuel Potter. Improvements in fluid actuated engines. 10.4.09.

(P.) 179/09. Henry Samuel Potter. Power device applicable to the actuation of jig conveyors and other apparatus more particularly in mining works. 10.4.09.

(P.) 180/09. Ernest Holroyd Martin. Improvements in the preparation of ore and the like for milling. 10.4.09.

(P.) 181/09. Charles H. Baker. An improved automatic jockey or rope grip for mechanical haulage. 14.4.09.

(P.) 182/09. Neil Kerr Webster. Improvements in apparatus for extracting metals from solution. 15.4.09.

(P.) 183/09. Hendrik Christensen. Improvements in the manufacture of bricks and the like. 15.4.09.

(C.) 186/09. Ernest Richard Royston (1), Harry Tee (2). Improvements in or connected with the manufacture of salt. 16/4/09.

(P.) 187/09. John Penhale (1), Harry Ormes Mascall (2). Process for disintegrating diamondiferous ground. 17.4.09.

(P.) 188/09. Robert Hutchinson Anderson. Improvements in rock drills. 19.4.09.

(P.) 189/09. Hans Charles Behr. Improvements in stamp mill cams. 19.4.09.

(P.) 190/09. Hans Charles Behr. Improvements in ore stamps. 19.4.09.

(P.) 191/09. Frederick Walter Bawden. Electric fuse igniter. 20.4.09.

(P.) 192/09. William Philip. Safety device for reversible winding and hauling plants. 20.4.09.

(P.) 193/09. Auguste Carron (1), Frank Emley (2), Eugene Radu Rossetti (3). Preservative composition for coating wooden and other structures. 20.4.09.

(P.) 194/09. Auguste Carron (1), Frank Emley (2), Eugene Radu Rossetti (3). Fire resisting water proof paint. 20.4.09.

(P.) 195/09. Thomas Gilnagh Maher. Improvements relating to the charging of holes with explosives. 21.4.09.

(P.) 196/09. George Tom Cannon. Improvements in rope grips or jockeys for mechanical haulage and the like. 22.4.09.

(P.) 197/09. James Hosking. An improved process in the milling of metalliferous ores and other minerals and substances. 23.4.09.

(C.) 199/09. John Frederick Webb (1), Percy Tarbutt (2). Improvements in filtering apparatus for use in connection with the washing of slimes and for other similar purposes. 23.4.09.

(P.) 204/09. John Coatts Shaw. Improvements appertaining to stoppers for bottles and similar vessels. 27.4.09.

(C.) 205/09. William Henry Morgan (1), James Densley (2), Harry Gray (3), Henry James Walduck (4), George Geering (5). A new or improved process for an apparatus for the manufacture of coal, water or oil gas, or combinations or mixtures of the said gases. 27.4.09.

(P.) 206/09. William Archibald Shearer. Improvements in acetylene gas lamps. 27.4.09.

(P.) 208/09. Frederick Busching. Improvements in means for distributing mine residues or other similar materials upon a mine dump or other depositing site. 27.4.09.

(P.) 209/09. John Pitkethly Smith (1), John Bruce (2) Improvements in safety brake arrangements applicable to mine cages and other hoisting apparatus. 27.4.09.

(P.) 211/09. George Briscoe. A rock drilling machine. 29.4.09.

Changes of Addresses.

Member and Associates are requested to notify the Secretary immediately of any change in address, otherwise it is impossible to guarantee the delivery of Journals or Notices. The Secretary should be at once notified of non-receipt of Journals and Notices.

ALLEN, ROBT., *l/o* Chihuahua; Apartado 2554, Mexico, D.F.
ARKELL, D. J., *l/o* Klerksdorp; P. O. Box 1865, Johannesburg.
BRADING, T., *l/o* Benoni; Vectis Reef, P. O. Colleen Bawn, Gwanda, Rhodesia.
COGLE, C. T., *l/o* Johannesburg,: Van Ryn G. M. Co., Ltd., P. O. Box 22, Benoni.
COMRIE, A., *to* Rosalind Mine, Essexvale, Rhodesia.
COOMBE, M. H., *l/o* Johannesburg: Consolidated Langlaagte G. M. Co., Ltd., P. O. Box 15, Langlaagte.
CROPPER, C. H., *l/o* London; Village Deep, Ltd., P. O. Box 1145, Johannesburg.
DANIELS, IVOR C., *l/o* Queque; Rose of Sharon and Shamrock Gold Mines, Ltd., Gwelo, Rhodesia.
FERGUSSON, MALCOLM, *l/o* Johannesburg; P. O. Box 164, Germiston.
FERRIS, V. GRINDLEY, *l/o* Ferreira Deep, Ltd.; Consolidated Gold Fields of South Africa, Ltd., P. O. Box 1167, Johannesburg.
GOSNEY, PERCY, *l/o* West Nicholson: Crocodile Reef, Essexvale, Rhodesia.
HALFORD, J. E., *l/o* Nigel: Midas Deep G. M. Co., Ltd., Randfontein.
HOLFORD, W. G., *l/o* Johannesburg; Apex Mines (Gold), Ltd., Benoni.
LARET, H., *l/o* Brakpan; Zeerust Lead Proprietary Mines, Ltd., P. O. Box 44, Zeerust.
LATHBURY, F. H., *l/o* Matopos; P. O. Box 213, Bulawayo.
LINDBERG, B., *l/o* Sumatra; "Mutual Life" Office, Helsingfors, Finland.
MANCE, J. C., *l/o* Sandy; c/o Messrs. Guthrie & Co., Ltd., Singapore.
MANNHEIM, H. C., *to* P. O. Box 585, Bulawayo, Rhodesia.
MARQUARD, J. D., *l/o* Rose Deep, Ltd.: Glen Deep, Ltd., P. O. Box 184, Germiston.
MARSHALL, C. C., *l/o* Boksburg: Geduld Proprietary Mines, Ltd., P. O. Box 41, Springs.
McMILLAN, Q. C., *to* P. O. Box 278, Germiston.
MITCHELL, A., *l/o* Surprise Siding; Dunraven Mine, Selukwe, Rhodesia.
POTTER, H. S., *to* 32, Plein Street, Johannesburg.
RAND, E. T., *l/o* East Rand; P. O. Box 9, Boksburg North.
RYAN, W. A., *to* Lonely Reef, Inyati, via Bulawayo.
SCAER, VAL. E., *l/o* Matopos; P. O. Box 69, Bulawayo.
SHORT, ARTHUR R., *l/o* Klerksdorp; Cinderella Deep, Ltd., P. O. Box 75, Boksburg.
SIM, A. M., *l/o* Cleveland; City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg.
SUMMERS, E., *to* P. O. Box 94, Klerksdorp.
THOMAS, H. MUSSON-, *l/o* Cleveland; City and Suburban G. M. Co., Ltd., P. O. Box 1026, Johannesburg.
THOMAS, WM., *l/o* Redruth; Champion Reef G. M. Co., Mysore State, S. India.
THORBURN, J. M., *to* P. O. Box 3427, Johannesburg.
WHYTE, P. H. M., *l/o* East Rand; P. O. Box 679, Johannesburg.
WOLVEKAMP, H., *to* Ketahoen, Benkoelen, Sumatra.

INDEX.

INDEX TO VOL. IX.

Compiled by the Secretary (Mr. Fred. Rowland).

Those Portions in Italics refer to Original Papers read before the Society.

PAGE

A

Abrest, E. Kohn—Analysis of Aluminium Powder 309
Absorption of Gold Amalgam on Copper Plates E. Halse 62
—— of Gold by Copper Plates E. Halse and W. E. Thomae 214
—— of Water by Coke 314
Abstracts, Chemistry 26, 52, 98, 128, 168, 212, 242, 276, 309, 365, 408, 435
—— Metallurgical 30, 55, 100, 130, 171, 214, 244, 278, 312, 368, 411, 437
—— Mining 34, 63, 101, 135, 173, 218, 245, 280, 315, 369, 412, 444
—— Miscellaneous 65, 102, 136, 175, 218, 248, 284, 320, 370, 414, 448
Accidents in Winding G. H. Winstanley 63
—— Mining, in the United States 246
Accounts, Statement of 423
Accumulation and Absorption of Gold Amalgam on Copper Plates E. Halse .. 62
Accuracy of Coal Analyses... 132
Acetate, Lead, in Cyanidation C. M. Eye 368
Acetylene in the Laboratory 169
Acid, Arsenious, The Theory and Practice of the Iodometric Determination of E. W. Washburn 27
—— Hippuric, Characteristic Test for W. M. Dean 243
—— Reaction of Mine Waters 176
—— Succinic 243
—— Tungstic, Determination of J. B. Ekeley and G. D. Kendall, Jr. ... 98
Action of Gases on Boiler Plates C. E. Stromeyer 320
—— of Hydrogen Sulphide on Alkaline Solutions of Zinc L. W. McCay 98, 128
Adair, A.—On Adair-Usher Process 50, 118
—— On Suggestions for a New Atomic Theory 433
—— *Usher Process A. Adair Discussion* 23, 48, 94, 118
Reply to Discussion 158
Adams, W. J.—Hints on Amalgamation and the General Care of Gold Mills (Review) 179
Address, Inaugural, President's (R. G. Bevington)... 2
Addresses, Changes of 36, 68, 104, 140, 180, 220, 252, 288, 324, 372, 416, 452
Admission of Associates 1, 37, 69, 105, 141, 181, 221, 254, 289, 325, 417
—— of Students 69, 141, 325, 373
Agriculture and Agricultural Chemistry ... 5
Air, Arsine in, Detection of Hebert and Heim 214
—— Currents in Mines 245
—— Dust in, Quantitative Estimation of Prof. M. Hahn 34
—— Hammer Drills, Merits and Demerits of G. E. Wolcott 35
—— Mine, Test for Purity of Dr. A. Smith 311

PAGE

Air, Moist, Influence of, on Quicklime James Gray 396
Alcohol Denaturant 30
Alexander, F. F.—On Gold Law Regulations 289, 290
—— On Effect on Amalgamation of Different Intervals of Time Between the Dressings of Plates 429
Alkali Destroys Cement 65
Alkalies from Magnesium, Separation of Browning and Drushel 30
Alkaline Solution, Titration of Zinc in E. B. van Osdel 28
—— Solutions of Zinc, Action of Hydrogen Sulphide on L. W. McCay 98, 128
All-Sliming Process in New Zealand ... 217
Alloy, Non-Expansive (Invar) 314
Alloys, Cobalt Tin 312
—— of Gold and Tellurium T. K. Rose ... 55
—— Qualities of 444
—— Sparks from A. von Welsbach ... 315
Alumina for Polishing Metals 244
Aluminium Powder, Analysis of E. Kohn-Abrest 309
Amalgam, Gold, on Copper Plates, Accumulation and Absorption of E. Halse ... 62
—— Platinum 30
Amalgamating Plates, The Silver Coating of W. A. Caldecott 142, 222
Amalgamation, The Economics of J. H. Haynes 438
—— *Effect on, of Different Intervals of Time between the Dressings of Plates G. O. Smart* 425
Discussion 427
Ammonia, Liquid F. W. Frerichs ... 367
—— Production of, from Atmospheric Nitrogen by means of Peat H. C. Woltereck 243
Ammonium Phosphomolybdate in the Determination of Phosphorus in Iron and Steel G. Chesneau 53
Amount of Coal Dust in Airways H. M. Hall 315
Analyses, Coal, Accuracy of 132
—— of S.A. Coals... 330
—— of Tool Steel 314
Analysis, Coke Hinrichsin and Taczak ... 412
—— of Aluminium Powder E. Kohn-Abrest 309
—— of Ashes and Alloys L. Parry (Review) 67
—— of Lead and Silver in Ores Dr. J. Lövy 169
—— of Lead Slag E. W. Buskett ... 99
—— of Mexican Gold-Siver Ores 10
—— of Pig Lead E. W. Buskett ... 129
—— Qualitative, A Rapid Method of W. P. Pollard 242
—— Volumetric, Titanous Sulphate as a Reducer of Iron in H. D. Newton 52
Anderson, T. S.—Direct Extraction of Copper from its Ores by Electrical Methods ... 216

PAGE

Andrews, A. J.—South African Coals and Their Economics 330
Discussion 391, 432
Ankylostomiasis in South Africa G. A. Turner 175
Annadale Copper Co. 216
Annual Dinner ... .289, 325, 352
—— Report of Council 418
Apparatus, Rescue 35
Appliances, Rescue: Lessons from Glencoe H. Kestner Discussion ... 21, 41
Reply to Discussion 42
Apprenticeship to Mining 4
Argall, P. — Historical Summary of the Cyanide Process 55
Argon, Extraction of Fischer and Ringe .. 130
Arseniated Hydrogen Poisoning 219
Arsenic in Ores, Rapid Method for the Estimation of H. E. Hooper ... 26
—— White, Dangers of 314
Arsenious Acid, The Theory and Practice of the Iodometric Determination of E. W. Washburn 27
Arsine in Air, Detection of Hebert and Heim 214
Asbestos, The Dressing of 441
—— Products, Classification of 442
Assay of Slimes Residues S. H. Pearce ... 401
—— —— —— A. Whitby.. 401
—— of Telluride Ores G. T. Holloway and L. E. B. Pearse 28
—— Valuations, Conversion Tables for W. J. Sharwood 320
Assays Bullion 185
—— *Grading, and Grinding Efficiencies Arthur Yates* 187
Discussion ... .238, 346, 395
Assaying, Chemicals used for, The Existence of Platinum Metals in J. G. Rose 168
—— Manual of Practical H. van F. Furman (Review) 450
—— *Routine, on a Westralian Mine W. B. Blyth* 184
Discussion 347, 393
Correspondence 435
Associates, Admission of ... 1, 37, 69, 105, 141, 181, 221, 254, 289, 325, 417
Atom, Structure of the Sir O. Lodge ... 29
Atomic Theory, Suggestions for a New Dr. J. Moir 334
Discussion 341, 392, 433
—— Weights, Exact Relations among the D. F. Comstock 55
—— Weights, 1909. Report of the International Committee on 310
Auerkieff, W. D.—Dissolution of Metallic Gold in Hydrochloric Acid in the presence of various organic substances ... 214
Austin, L. S.—Metallurgy of Common Metals (Review) 219
Australian Mining and Engineering Review... 323
Automatic Estimation of CO_2 in Furnace Gases Dr. C. A. Keane and Dr. H. Burrows 54
—— Filter Press R. D. Lemon and R. P. Holmes 244

B

Bacteria as Agent in the Oxidation of Amorphous Carbon M. C. Potter ... 138
Balance Sheet, 1908-9 423
Bantjes Consolidated Mine 216

PAGE

Barium, Chloride of, for Hardening Steel ... 101
—— Determination of 99
Batteries, Small, Oil Engines for W. F. Smith 32
Battery and Cyanide Gold Smelting A. Thomas 6
Discussion 120, 162
Do. Reply to 191
Correspondence 208
Addendum 418
—— Screen Samples 189
Baubigny, H.—Separation of Silver Chloride and Iodide 130
Bauer and Heyn.—The Corrosion of Iron ... 212
—— —— Rusting of Iron 366
Bauman Auto-Sulphur Printing Process ... 217
Beaupre, M. Mauricheau.—New Method of Making Hydrogen 243
Bell, H. C. F.—On Theory of Blasting with High Explosives 203
Bending Stresses in Wire Ropes 318
Bettel. W.—Purification of Mercury ... 214
Bevington, R. G. (President). — *Inaugural Address* 2
—— On a Few Notes on Stamp Milling ... 21
—— On Alteration of Rules for Students 293, 352
—— On and at Annual Dinner 289, 353, 354, 365
—— On Annual Elections 374, 424
—— On Annual Report 422
—— On Battery and Cyanide Gold Smelting 9, 51, 123, 163, 418
—— On Cheap Platinum Parting Apparatus 257
—— On Christmas Greetings 207
—— On Coloured Explosives 374
—— On Coloured Wrappers for Explosives ... 292
—— On Covers for Extractor Boxes .. 326
—— On Counterweighted Blind for Sands Collecting Vat 327
—— On Cupelling Gold-Lead Bullion .. 158
—— On Distribution of the Gold produced on the Rand 432
—— On Effect on Amalgamation of different intervals of time between the Dressing of Plates 427, 429
—— On Election as Corresponding Member of Council by I.M.M. 254
—— On Estimation of Carbon Monoxide .. 265
—— On First Meeting in Transvaal University College 253
—— On Fused Silica Ware 292
—— On Gases Resulting from the Use of High Explosives 151
—— On General Business 221
—— On Gold Crystals 183
—— On Gold Law Regulations 290
—— On Grading Assays and Grinding Efficiencies 191
—— On Illness of Council Members ... 255
—— On Improved Glasses for Testing for Potassium 293
—— On Local Sections 142, 209
—— On Mexican Institute of Mining and Metallurgy 424
—— On Mine Disasters 254
—— On Mines Trials Committee ... 37, 69, 254
—— On New Year Greetings 239
—— On Note on a Problem during Shaft Sinking 73, 305
—— On Notes on Rand Mining ... 21, 94
—— On Ore Treatment at the Giant Mine ... 156
—— On Patio Process 110
—— On Pilgrims Rest Gold Fields 300

PAGE
Bevington, R. G. (President)—On Precipitation of Gold from Solution by Carbonaceous Matter 328, 329
—— On Precipitation from Cyanide Solutions by Zinc Shavings and Dust 222
—— On Reminiscences of the Early Rand 38, 97, 125
—— On Rescue Appliances: Lessons from Glencoe 42, 45
—— On Routine Assaying on a Westralian Mine 186
—— On Silver Coating of Amalgamating Plates 143
—— On Simmer Deep Visit ... 350, 425
—— On Small Mines of Rhodesia 81
—— On Sociable Intercourse 142
—— On Some Features of Silver Ore Treatment 13
—— On S. A. Coals and their Economics 333, 391
—— On Stope Measurements 388
—— On Suggestions for a New Atomic Theory 341
—— On Technical Societies Common Room ... 255
—— On Theory of Blasting with High Explosives 118, 196, 202, 203
—— On Visit to Premier Mine 181
—— On Vote of Thanks to Secretary ... 2
—— On War Memorial 422
—— On Zinc Dust Precipitation .. 75, 76
—— Replying to Vote of Thanks 6
—— Returning Thanks for Council .. 425
—— Valedictory 424
Blair, A. A.—The Chemical Analysis of Iron (Review) 251
Blast Furnace Slag, Cement from 30
Blasting Gelatine 317
—— Rounds, Results obtained from ... 145
—— *with High Explosives, Theory of E. M. Weston* 110
Discussion 193, 232
Do. Reply to 343
Bleaching Powder 53
Blown Out Shots D. Foster 319
Blyth, W. B.—Routine Assaying on a Westralian Mine 184
Discussion 347, 393
Correspondence 435
Boiler Feedwater and its Treatment ... 408
—— Plates, Action of Gases on C. E. Stromeyer 320
Boiling Points of Metals Dr. O. P. Watts 100
—— Point of Sulphur, Effect of Pressure on the Dr. J. A. Harker and F. P. Sexton 249
Bölling, F.—Silundum 436
Books, New, Reviews and 35, 66, 102, 139, 179, 219, 249, 286, 322, 371, 414, 450
Borehole, The Deepest, in the World ... 447
Boreholes, Water in, Determining Depth of C. B. Burdick 64
Bradford, Wager—On Precipitation of Gold from Solution by Carbonaceous Matter 329
Brett, H. T.—On Coloured Wrappers for Explosives... 240
Bricks, Fire, Chromite for M. Simonis . 217
Briquetting Precipitates 443
Brown, A. Selwyn—Macquisten Process of Flotation 411
Browne, F.—The Estimation of Graphite ... 128
Browning, W. C.—The Carrying Capacity of Launders 441
—— and Drushel — Separation of Alkalies from Magnesium 30
Building of Explosive Works O. Guttmann 130
Bullion Assays 185
—— Bars, Pitted Gold 216
—— Gold, Silver in E. H. Taylor ... 170
—— *Gold-Lead, A Few Notes on Cupelling G. Melvill* 157
Discussion 158
Do. Reply to 345
—— *Gold Precipitates and, The Smelting of, with Oil Fuel A. Yates* 429
—— Re-exported from the United Kingdom 267
—— Retained in the United Kingdom ... 267
Burdick, C. B.—Determining Depth of Water in Boreholes 64
Burning of Coal without Smoke in Boiler Plants D. T. Randall 67
Burrows, Dr. H. and Keane, Dr. C. A.—Automatic Estimation of CO_2 in Furnace Gases 54
Burt's Slimes Filter 172
Buskett, E. W.—Analysis of Lead Slag ... 99
—— Analysis of Pig Lead 129
Bye-Products 8

C

Caisson Disease 317
Calcining Zinc-Gold Slimes 7
Calcium Hypochlorite, Stable 53
Caldecott, W. A.—Precipitation of Gold from Solution by Carbonaceous Matter 327, 329, 330
—— *Silver Coating of Amalgamating Plates*... 142
—— *Some Features of Silver Ore Treatment in Mexico—Reply to Discussion*... ... 10
—— *Correspondence* 51, 97, 307
—— *Use of the Vacuum Pump in the Cyaniding of Sand* 240
—— On Capacity of Circular Vats per foot of Depth 127
—— Classification for Tube Milling ... 312
—— On Distribution of the Gold produced on the Rand 432
—— On Local Sections 209
—— On Ore Treatment at the Giant Mine 156, 157
—— On Patio Process 110
—— On Pilgrim's Rest Gold Fields... ... 300
—— On Reminiscences of the Early Rand ... 123
—— On Reply to vote of thanks 13
—— On Researches upon the Telluride Gold Ores of Cripple Creek, Colorado ... 434
Callow Screen, The 313
Calorific Deterioration in Stored Coal ... 34
Cam Shafts, Cushioning Vibrations of A. del Mar 411
Capacity of Circular Vats per Foot of Depth W. A. Caldecott 127
Capacity of Launders, Carrying W. C. Browning 441
—— of Pipes 320
Carat, The 66
Carbon, Amorphous, Bacteria as Agent in the Oxidation of M. C. Potter ... 138
—— Dioxide in Furnace Gases, Automatic Estimation of Dr. C. A. Keane and Dr. H. Burrows 54
—— —— in Mines 245
—— for Diamond Drill Bits, The Selection of 34
—— Monoxide and Carbon Dioxide, Tests for 245
Carbon Monoxide in Mine Gas, The Estimation of Dr. E. H. Weiskopf 258
Appendix 306

PAGE
Carbonaceous Matter, Precipitation of Gold from Solution by W. A. Caldecott 327, 374, 400
Carbonisation at Low Temperatures V. B. Lewes 29
Carnegie, A.—Conservation of Iron Ore and Coal Resources 248
Carrying Capacity of Launders W. C. Browning 441
Carter, T. Lane —On Zinc Dust Precipitation 207
Castings, Steel, Vanadium in 61
Cement, Alkali destroys 65
—— from Blast Furnace Slag 30
—— Iron Ore. in Sea Water 178
—— Micrographical Study of 321
Centrifugal Mine Pumps 413
Changes of Addresses, 36, 68, 104, 140, 180, 220, 252, 288, 324, 372, 416, 452
Characteristic Test for Hippuric Acid W. M. Dean 243
Charlton, A. G.—Report Book for Mining Engineers (Review).. 103
Cheap Platinum Parting Apparatus Prof. G. H. Stanley 256
Discussion 257
Chemical Analysis of Iron A. A. Blair (Review) 251
—— Products, Hard 311
Chemicals used for Assaying, The Existence of Platinum Metals in J. G. Rose... 168
Chemistry Abstracts, 26, 52, 98, 128, 168, 212, 242, 276, 309, 365, 408, 435
—— Geo- F. W. Clarke 448
Chesneau, G.—Ammonium Phosphomolybdate in the Determination of Phosphorus in Iron and Steel 53
Chloride of Barium for Hardening Steel ... 101
Chloroplatinate, Potassium, Solubility of G. S. Jamieson, L. H. Levy and H. L. Wells 100
Chromite for Fire Bricks M. Simonis ... 217
Chromium, A new form of 29
Chudleigh, C. (The Mayor) at Annual Dinner 365
Cinderella Deep Mine 216
Cinnabar 98
Circular Vats, Capacity of, per Foot of Depth W. A. Caldecott 127
Circular Shafts 446
Clark, A. J.—Precipitation from Cyanide Solutions by Zinc Shavings and Dust: A Comparison of Results and Costs ... 222
Clarke, F. W.—Geo-chemistry 448
Classification for Tube-Milling W. A. Caldecott 312
Clean-up at the Portland Mill, Cyanide Precipitation and J. M. Tippett ... 31
Closer Co-operation between Technical Societies 421
Coal Analyses, Accuracy of 132
—— and Wood, Ignition Points of 134
—— Deterioration of, by Weathering C. Scholtz 65
—— Dust as a Factor in Mine Explosions H. H. Payne 174
—— —— Explosions M. Dautriche ... 63
—— —— —— Prevention of M. Dautriche 173
—— —— Explosive Properties of H. M. Hall 315
—— —— Ignition J. Neal 218
—— —— in Mines W. E. Garforth ... 246
—— Mining D. Burns and G. L. Kerr (Review) 322
PAGE
Coal Mining, Dust in B. F. Jones ... 102
—— Resources and Iron Ore, Conservation of A. Carnegie 248
—— Spontaneous Combustion of Erdmann and Stolzenberg 65
—— Stored, Calorific Deterioration of ... 34
Coals, South African, and their Economics A. J. Andrews 330
Discussion 391 432
Cobalt and Nickel, Detection of, when present together H. Grossmann and W. Heilbron 129
—— Ores, Silver-, Sampling of A. A. Cole 172
—— Tin Alloys S. Shemtschushny and S. Bielynski 312
Coehn, A., and Jacobson, C. L.—Passivity of Gold 32
Cohen, Dr. J. B.—Organic Chemistry (Review) 35
Coke, Absorption of Water by 314
—— Analysis Hinrichsin and Taczak ... 412
Cole, A. A.—Sampling of Silver-Cobalt Ores 172
Collieries, Electrical Equipment of W. G. Duncan and D. Penman (Review) ... 323
—— Mechanical Engineering of T. C. Futers (Review) 323
Collings, B. I.—Small Mines of Rhodesia ... 76
Discussion... ... 126, 166, 206, 275
Colorimetric Determination of Dissolved Oxygen in Water G. B. Frankforter, G. W. Walker and A. D. Wilmot ... 311
—— Methods T. Ewan... 366
Coloured Explosives 373
Coloured Wrappers for Explosives (Correspondence, etc.) 240 291
Combellack, J. H. (Obituary) 167
Combustion of Coal, Spontaneous Erdmann and Stolzenberg 65
Common Room, Technical Societies... ... 255
Competitor of the Electric Furnace 440
Compound Building, a New Type of Native, of all Metallic Construction. C. B. Kingston—Discussion 22
Reply to Discussion 81
Compressed Air Plant for Mines Prof. R. Peele (Review) 219
Computing Ore Values, On certain Errors in H. H Knox... 135
Comstock, D. F.—Exact Relations among the Atomic Weights 55
Concentrates carrying Copper, Treatment of B. L. Gardiner 130
Concrete Piles, in Mills 98 128
—— Reinforced M. Kahn 178
Conservation of Iron Ore and Coal Resources A. Carnegie 248
Contributions and Correspondence, 24, 51, 97, 127, 268, 239, 307, 400, 435
Control of Disinfectants, The T. H. Lloyd 310
Conversion Tables for Assay Valuations W. J. Sharwood 320
Cooke, R. C. H.—Description of Ore Treatment at the Giant Mine, Hartley District, Rhodesia 162
Discussion 156
Coombe, M. H.—On Notes on Rand Mining... 48
—— *Reminiscences of the Early Rand* ... 38
Discussion 97, 123, 204, 227, 272
Do. Reply to 273
Copper and Nickel, Electrolytic Separation of A. Thiel 53

PAGE
Copper, Direct Extraction of, from its Ores by Electrical Methods E. Spargo . . 216
—— Iodometric Determination of F. A. Gooch and F. H. Heath 30
—— Plate Absorption E. Halse and W. E. Thomae 214
—— Plates, Accumulation and Absorption of Gold Amalgam on E. Halse .. 62
—— Process, The Laszczynski 62
—— Slimes, Electro Metallurgical Treatment of D. Hollis and others 30
—— Treatment of Concentrates carrying B. L. Gardiner 130
—— Volumetric Estimation of, as Cuprous Thiocyanate by Potassium Iodate G. S. Jameson, H. L. Levy and H. L. Wells 54
Correspondence and Contributions, 24, 51, 97, 127, 239, 268, 307, 400, 435
Corresponding Member of Council, appointment of President as, by Institution of Mining and Metallurgy 254
—— Members of Council, ... 2 5
Corrosion in Steel, Sulphur and G. N. Huntly 437
—— of Iron Heyn and Bauer 212
—— —— Dr. W. H. Walker 435
Costs, Milling R. S. Handy 368
Cotter, A. J.—On Reminiscences of the Early Rand 231
Council, Annual Report of 418
—— Corresponding Members of ... 2 5
—— Election of Officers and 424
—— Meetings, Attendances at 420
Counterweighted Blind for Sands Collecting Vat C. W. Dowsett 326
Covers for Extractor Boxes K. L. Graham 325
Crane, W. R.—Gold and Silver (Review) ... 179
Cripple Creek Dumps, Cyaniding 437
Croghan, E. H.—On Adair-Usher Process ... 119
—— On a more Sociable Feeling 141
—— On Precipitation of Gold from Solution by Carbonaceous Matter ... 328, 329
Crookes, Sir W.—Use of Iridium and Rhodium Crucibles 217
Crowe, T B (edited by).—Researches upon the Telluride Gold Ores of Cripple Creek, Colorado 398
Discussion 434
Crucibles, Iridium and Rhodium, Use of Sir W. Crookes 217
—— Treatment of New Plumbago 7
Crushing, Ore 62
—— *Plants, Wet-, Westralian, with some Notes on Labour Efficiency G. W. Williams Discussion* 24
Reply to Discussion 270
Crystallisation of Shafts, To Prevent F. S. Kirkland 134
Crystals, Gold A. McA. Johnston ... 182
Cullen, W.—The Gases Resulting from the Use of High Explosives 144
Discussion 235, 274, 306
Cullen, W.—On Annual Elections 374
—— On Coloured Explosives 373
—— —— Wrappers for Explosives 291
—— On Diameter of Explosives Cartridges ... 291
—— On Fused Silica Ware 292
—— On Precipitation of Gold from Solution by Carbonaceous Matter 374
—— On South African Coals and their Economics 391
PAGE
Cullen, W.—On Theory of Blasting with High Explosives 118
—— Vote of Thanks to President 6
Cupellation Temperatures, Determination of Prof. C. H. Fulton 28
—— —— Silver Prof. C. H. Fulton .. 435
Cupelling Gold-Lead Bullion, A Few Notes on G. Melvill 157
Discussion 158
Do. Reply to 345
Curtis, J. S.—On Reminiscences of the Early Rand 123
Cushioning Vibrations of Cam Shafts A. del Mar 411
Cyanidation, Lead Acetate in C. M. Eye 368
—— of Silver Ores D. Mosher 437
Cyanide Gold Smelting, Battery and A. Thomas 6
Discussion 120, 162
Do. Reply to 191
Correspondence 208
Addendum 418
—— Manufacture of P. de Wilde ... 239
—— Precipitation and Clean-up at the Portland Mill J. M. Tippett 31
—— Process E. B. Wilson (Review) ... 179
—— —— Historical Summary of the P. Argall 55
—— *Solutions, Precipitation from, by Zinc Shavings and Dust A. J. Clark* ... 222
—— Treatment, Settling Slimes in 411
—— —— —— —— H. G. Nichols ... 56
—— Vats, Protective Coatings for W. H. Gaze 61
Cyanides, Sulpho- and Ferro-, Determining L. M. Green 213
Cyaniding Cripple Creek Dumps 437
—— *of Sand, The Use of the Vacuum Pump in the W. A. Caldecott* 240

D

Dangers of White Arsenic 314
Darling, G. A.—On Reminiscences of the Early Rand 227
Dautriche, M.—Coal Dust Explosions ... 63
—— Prevention of Coal Dust Explosions ... 173
Davis, W. H.—Microscopic Examination of Roasted Sulpho Telluride Ores ... 62
Dawe, W. H.—At Annual Dinner 356
Dean, W. M.—A Characteristic Test for Hippuric Acid 243
Deep Mining 64
Deepest Borehole in the World, The ... 447
Denaturant, Alcohol 30
Description of Ore Treatment at the Giant Mine, Hartley District, Rhodesia R. C. H. Cooke 152
Design and Equipment of Small Chemical Laboratories R. K. Meade (Review) 103
Detection of Arsine in Air Hebert and Heim 214
—— of Mercury in Nitro-Glycerine Compounds E. A. Mann .. 214
—— of Nickel and Cobalt when present together H. Grossmann and W. H. Heilbron 129
—— of Tellurium 212
Deterioration of Coal by Weathering C. Scholtz 65
—— of Stored Coal, Calorific - ... 34
Determination of Cupellation Temperatures C. H. Fulton 28

PAGE
Determination of Nickel 100
—— of Succinic Acid in presence of Fixed Acids in Wines and Fermented Liquids E. Pozzi-Escot 243
—— of Tin in Tinstone Koomans ... 30
—— of Tungstic Acid J. B. Ekeley and G. D. Kendall, Jr. 98
—— of Vanadium A. M. Wilson ... 52
Determining Depth of Water in Boreholes C. B. Burdick 64
—— Sulpho- and Ferro-Cyanides, &c. L. M. Green 213
Diameter of Explosive Cartridges 291
Diamond, Origin of the 367
Dinner, Annual 289, 325, 352
Direct Extraction of Copper from its Ores by Electrical Methods E. Spargo .. 216
Disasters at Mount Morgan, Report on ... 315
Disease, Caisson 317
Disinfectants, The Control of T. H. Lloyd 310
Dissolution of Metallic Gold in Hydrochloric Acid in presence of various organic substances W. D. Auerkieff ... 214
Distribution of the Gold Produced on the Rand Dr. T. K. Rose 265
Discussion 349 432
Ditte, A.—Sulphides and Double Sulphides ... 98
Dos Estrellas, Treatment of the Gold and Silver Precipitates at W. Neal . 443
Dovan, J.—Estimation of Gold and Palladium in Solution.. 171
Dowling, W. R.—On Zinc Dust Precipitation 76
—— On Cupelling Gold-Lead Bullion ... 158
—— On Counterweighted Blind for Sands Collecting Vat 327
—— On Effect on Amalgamation, &c. ... 428
Dowsett, C. W.—Counterweighted Blind for Sands Collecting Vat 326
—— *Low Fall Sands Return Contrivance* ... 326
Dressing of Asbestos, The 441
Drill Bits, Diamond, The Selection of Carbon for... 34
Drilling, Record Speed in 246
Drills, Air-Hammer, Merits and Demerits of G. E. Wolcott 35
—— Rock, Dust Collector for M. Rieumaux 64
Drum Lummon Mine 214
Drushel and Browning.—Separation of Alkalies from Magnesium 30
Dry Air Blast in Steel Making P. Longmuir 217
Duarte, C. Perez.—The Patio Process ... 105
Discussion 110
Dumps, Cyaniding Cripple Creek 437
Duncan, W. G., and Penman, D.- Electrical Equipment of Collieries (Review) ... 323
Durability of Modern Electrical Apparatus ... 138
Durnerin, M.—Rise of Earth Temperature ... 101
Dust, Coal, in Mines W. E. Garforth ... 246
—— Collector for Rock Drills M. Rieumaux 64
—— Explosions, Coal M. Dautriche 63
—— —— Prevention of Coal M. Dautriche 173
—— Ignition, Coal J. Neal 218
—— in Air, Quantitative, Estimation of Prof. M. Hahn 34
—— in Coal Mining B. F. Jones ... 102
—— Zinc, Zinc in De Koninck 311
Dwight and Lloyd's Sintering Process ... 32
Dynamite Factory, Visit to 105, 151, 168

PAGE
E
Early Rand, Reminiscences of the M. H. Coombe 38
Discussion 97, 123, 204, 227, 272
Do. Reply to 273
Earth Temperature, Rise of M. Durnerin 101
Economics of Amalgamation, The J. H. Haynes 438
—— of Tube Milling H. W. Fox ... 57
Edgar, G.—Separation of Vanadium and Molybdenum 52
Edwards, A. L.—On Reminiscences of the Early Rand 231
—— On Small Mines of Rhodesia 275
Effect of Pressure on the Boiling Point of Sulphur Dr. J. A. Harker and F. P. Sexton 249
—— *on Amalgamation, of Different Intervals of Time between the Dressings of Plates G. O. Smart* 425
Discussion 427
Efficiency, Labour, Westralian Wet-Crushing Plants, with some Notes on G. W. Williams. Discussion 24
Reply to Discussion 270
Ekeley, J. B., and Kendall, Jr., J. D.—Determination of Tungstic Acid ... 98
Elastic Iron Pit Props 218
Election of Members 1, 37, 69, 105, 141, 181, 221, 254, 289, 325, 373, 417
—— of Officers and Council 424
Electric Furnace, Competitor of the ... 440
—— —— its Evolution, Theory and Practice Dr. A. Stansfield (Review) 102
—— Lighting in Glasgow 219
Electrical Apparatus, Durability of Modern... 138
—— Equipment of Collieries W. G. Duncan and D. Penman (Review) ... 323
—— Methods, Direct Extraction of Copper from its ores by E. Spargo ... 216
Electrically Driven Mine Pumps S. F. Walker 412
Electro Metallurgical Treatment of Copper Slimes D. Hollis, F. P. Lannon, T. W. Quayle and P. D. Grommon .. 30
—— Metallurgy J. B. C. Kershaw (Review) 102
Electrolytic Separation of Nickel and Copper A. Thiel 53
Engines, Oil, for Small Batteries. W. F. Smith 32
Erdmann and Stolzenberg.—Spontaneous Combustion of Coal 65
Erratum 309
Estimation of Carbon Monoxide in Mine Gas Dr. E. H. Weiskopf 258
Appendix 306
—— of Gold and Palladium in Solution J. Dovan 171
—— of Graphite F. Browne 128
—— of High Temperatures Dr. F. W. Skirrow 134
Eucalyptus for Mine Timbers L. W. Symmes 316
Evans, Lewis—On Diameter of Explosives Cartridges 292
—— On Coloured Explosives 373
Ewan, T.—Colorimetric Methods 366
Exact Relations among the Atomic Weights D. F. Comstock 55
Existence of Platinum Metals in Chemicals used for Assaying J. G. Rose ... 168

PAGE
Experiment in Mine Ventilation 174
Explosions, Coal Dust M. Dautriche .. 63
—— —— —— Prevention of M. Dautriche 173
—— Mine C. B. Ross 247
—— —— Coal Dust as a Factor in H. H. Payne 174
Explosive Properties of Coal Dust H. M. Hall 315
—— Works, Building of O. Guttmann .. 130
—— Advantage of Blasting Gelatine as an .. 317
Explosives—Blown Out Shots D. Foster... 319
—— Cartridges, Diameter of 291
—— Coloured 373
—— Coloured Wrappers for (Correspondence, etc.) 240, 291
—— *High, The Gases resulting from the use of W. Cullen* 144
Discussion 235, 274, 306
—— *High, Theory of Blasting with E. M. Weston* 110
Discussion 193, 232
Do. Reply to 343
—— in Mining 447
—— to Destroy 318
—— Twenty Year's Progress in O. Guttmann 247
—— Use of A. H. Merrin 101
Extraction and Uses of Molybdenum ... 171
—— of Argon Fischer & Ringe 130
Extractor Boxes, Covers for K. L. Graham 325
Eye, C. M.—Lead Acetate in Cyanidation ... 368

F

Factories, The Ventilation of Dr. J. S. Haldane 136
Fatigue Test for Wire Ropes, Appliance for... 64
Fay, Dr. H.—Steel Rails 171
Feedwater, Boiler, and its Treatment ... 408
Ferro-cyanides, Determining Sulpho- and L. M. Green 213
Few Notes on Cupelling Gold-Lead Bullion, A G. Melrill 157
Discussion 158
Do. Reply to 345
—— —— *on Stamp Milling, A W. H. Jane*
Discussion 21, 46
Do. Reply to 47
Filter-Press, Automatic R. D. Lemon and R. P. Holmes 244
—— Slimes H. M. Leslie 100
Filtration of Slimes H. G. Payne ... 172
Fine Grinding Results in New Zealand . 217
Fineness of Bullion, Table for 321
Fire Bricks, Chromite for M. Simonis ... 217
Fireproofing Wood 248
Fischer and Ringe—Extraction of Argon ... 130
FitzPatrick, Sir Percy—At Annual Dinner ... 357
Flotation, Macquisten Process of A. Selwyn-Brown 411
Flux for Melting Washings, etc., containing Iron 56
Fluxes 444
Foster, D.—Blown Out Shots 319
Four versus Five Stamp Battery 46
Fox, H. W.—Economics of Tube Milling ... 57
Fracture Planes, Drilling across 196
Frerichs, F. W.—Liquid Ammonia 367
Fuel Tests and Analyses 330
Fulton, Prof. C. H.—Determination of Cupellation Temperatures 28
—— Silver Cupellation Temperatures .. 435

PAGE
Furman, H. van F.—A Manual of Practical Assaying (Review) 450
Furnace, Electric, Competitor of the ... 440
Furnaces on a Westralian Mine 184
Further Notes on the Utilisation of Waste Heat in Slimes Treatment A. Salkinson... 308
Fuse, Safety, Testing of, by X Rays Jas. Thomas 183
—— Test E. A. Mann 175
Fused Silica Ware W. Cullen 292
Futers, T. C.—Mechanical Engineering of Collieries (Review) 323

G

Gardiner, B. L—Treatment of Concentrates carrying Copper 130
Garforth, W. E.—Coal Dust in Mines ... 246
Gas, Mine, The Estimation of Carbon Monoxide in Dr. E. H. Weiskopf 258
Appendix 306
Gases, Action of, on Boiler Plates C. E. Stromeyer 320
—— Furnace, Automatic Estimation of Carbon Dioxide in Dr. C. A. Keane and Dr. H. Burrows 54
—— *Resulting from the Use of High Explosives, The W. Cullen* 144
Discussion 235, 274 306
—— Volcanic 448
Gayley Dry Air Blast 217
Gaze, W. H.—Protective Coatings for Cyanide Vats 61
Geduld Proprietary Mine 216
Genesis of Metallic Ores and of the Rocks which enclose them B. Symons (Review) 179
Geo Chemistry F. W. Clarke 448
Geological Notes on Rhodesia 76
Geology of Coal and Coal Mining Dr. W. Gibson (Review) 139
—— of Pilgrims Rest 293
Giant Mine, Rhodesia, Description of Ore Treatment at the R. C. H. Cooke 152
Gibson, Dr. W.—The Geology of Coal and Coal Mining 139
Giessen, W.—Production of the Rare Metals 245
Glasgow, Electric Lighting in 219
Glasses for Testing for Potassium, Improved Dr. J. Moir 292
Gold Amalgam on Copper Plates, Accumulation and Absorption of E. Halse ... 62
—— and Palladium in Solution, Estimation of J. Dovan 171
—— and Silver W. R. Crane (Review) ... 179
—— —— Precipitates at Dos Estrellas, Treatment of the W. Neal 443
—— —— Volatilisation of Prof. Richards 33
—— and Tellurium, Alloys of T. K. Rose 55
—— Bullion Bars, Pitted 216
—— —— Silver in E. H. Taylor ... 170
—— *Crystals* A. McA. Johnston ... 182
—— Fine, and Silver, Volume and Weight... 321
—— it's Geological Occurrence and Geographical Distribution J. Malcolm Maclaren (Review) 450
—— Law Regulations 289
—— *Lead Bullion, a Few Notes on Cupelling G. Melrill* 157
Discussion 158
Do. Reply to 345
—— Metallic, Dissolution of, in Hydrochloric Acid in presence of various organic substances W. D. Auerkieff 214

PAGE
Gold Ores, Telluride, of Cripple Creek, Researches upon the *T. B. Crowe* ... 398
Discussion 434
—— —— Unsuspected Minerals in W. P. Headden 34
—— Passivity of A. Coehn and C. L. Jacobson 32
—— *Precipitates and Bullion with Oil Fuel, The Smelting of* *A. Yates* ... 429
—— *Precipitation of, from Solution by Carbonaceous Matter* *W. A. Caldecott* 327, 374, 400
—— *Produced on the Rand, The Distribution of the* *Dr. T. Kirke Rose* 265
Discussion 349, 432
—— Retained on Copper Plates 214
—— Separation of, from Auriferous Platinum E. Schardin 56
—— *Smelting, Battery and Cyanide* *A. Thomas* 6
Discussion 120, 162
Do. Reply to 191
Correspondence 208
Addendum 418
Gooch, F. A., and Heath, F. H.—Iodometric Determination of Copper 30
Grading Assays and Grinding Efficiencies *A. Yates* 187
Discussion 238, 346, 395
Graham, K. L.—Covers for Extractor Boxes ... 325
—— On Adair-Usher Process 48
—— On Student Membership 352
—— On Zinc Dust Precipitation 232
Grain of Witwatersrand Strata 196
Graphite, Estimation of F. Browne ... 128
Gravitation, Theories of 449
Gray, James—The Influence of Moist Air on Quicklime 396
—— Vote of Thanks to Secretary ... 2, 422
—— Returning Thanks for Scrutineers ... 425
Green, L. M.—Determining Sulpho- and Ferro-Cyanides 213
Gregory, Prof.—What is a Mineral 414
Grinding Efficiencies, Grading Assays and *Arthur Yates* 187
Discussion 238, 346, 395
Grommon, P. D., and Others—Electro-Metallurgical Treatment of Copper Slimes... 30
Grossmann, H., and Heilbron, W.—Detection of Nickel and Cobalt when present together 129
Guanajuato Consolidated Mining and Milling Co.—Staff and Distribution of Labour for 1907 445
Guide to Technical Writing T. A. Rickard (Review) 250
Guides and Ropes, Lubricating 318
Gulliver, G. H.—Metallic Alloys (Review)... 323
Guttman, O.—The Building of Explosive Works 130
—— Twenty Years' Progess in Explosives ... 247

H

Hahn, Prof. M.—Quantitative Estimation of Dust in Air 34
Haldane, Dr. J. S.—On Gases Resulting from the Use of High Explosives 274
—— The Ventilation of Factories 136
Hall, H. M.—Amount of Coal Dust in Airways 315
Halse, E.—Accumulation and Absorption of Gold Amalgam on Copper Plates ... 62
Halse, E , and Thomae, W. E.—Copper Plate Absorption 214
Hamilton, E. M.—On Some Features of Silver Ore Treatment in Mexico 97
—— J. Laurie (Obituary) 208
Handy, R. S.—Milling Costs 368
Hard Chemical Products 311
Harker, Dr. J. A , and Sexton, F. P.—Effect of Pressure on the Boiling Point of Sulphur 249
Hartley, A H.—On Adair-Usher Process ... 94
—— On Battery and Cyanide Gold Smelting 162
Haulage at Pilgrim Rest 298
Haynes, J. H.—The Economics of Amalgamation 438
Headden, W. P.—Unsuspected Minerals in Gold Ores 34
Health Conditions Underground 259
Heat, An Indicator of Excessive 175
—— *Waste, in Slimes Treatment, Further Notes on the Utilisation of* *A. Salkinson* 308
Heath, F. W. and Gooch, F. A.—Iodometric Determination of Copper 30
Hebert and Heim.—Detection of Arsine in Air 214
Heilbron, W., and Grossmann, H.—Detection of Nickel and Cobalt when present together 129
Heim and Hebert.—Detection of Arsine in Air 214
Henry, E.—Mine Timber Preservation ... 63
Heymann, A.—On Rescue Appliances, &c. ... 41
Heyn and Bauer.—The Corrosion of Iron ... 212
—— —— Rusting of Iron 366
Hidden Treasure Mine 215
High Temperature 56
Hindley, J., and Stone, J.—Stresses on Winding and Conducting Ropes 318
Hinrichsin and Taczak.—Coke Analysis ... 412
Hints on Amalgamation and the General Care of Gold Mills.—W. J. Adams (Review) 179
Hippuric Acid, Characteristic Test for W. M. Dean 243
Historical Summary of the Cyanide Process P. Argall 55
Hoisting, Long Incline 86
—— Rock 15, 16
Holes, Average Length of Drill 14
Hollis, D., and others.—Electro-Metallurgical Treatment of Copper Slimes.. ... 30
Holloway, G. T., and Pearse, L. E. B.—The Assay of Telluride Ores 28
Holmes, R. P., and Lemon, R. D.—Automatic Filter Press 244
Homestake Slimes Plant,' Operation Costs of the Merrill Process at the 215
Hooks, Safety Sinking H. Louis... ... 136
Hooper, H. E.—A Rapid Method for the Estimation of Arsenic in Ores 26
Horan, P.—Tapered Timber Props 369
Huntly, G. N.—Sulphur and Corrosion in Steel 437
Hydrazine, Production of, from Inorganic Sources — Raschig 54
Hydrogen, New Method of Making M. M. Beaupre 243
—— Peroxide Cell... 169
—— Poisoning, Arseniated.. 219
—— Sulphide, Action of, on Alkaline Solutions of Zinc L. W. McCay 98 128
Hypochlorite, Stable Calcium 53

PAGE

I

Ignition Points of Wood and Coal 134
Illness of Council Members... 255
Improved Glasses for Testing for Potassium Dr. J. Moir 292
Inaugural Address, President's (R. G. Berington) 2
Influence of Moist Air on Quicklime James Gray 396
Ingle, H.—Mine Tailings on the Rand ... 177
Institution of Mining and Metallurgy ... 254
Iodide, Separation of Silver Chloride and H. Baubigny 130
Iodometric Determination of Arsenious Acid, The Theory and Practice of the E. W. Washburn 27
—— Determination of Copper F. A. Gooch and F. H. Heath 30
Iridium and Rhodium Crucibles, Use of Sir W. Crookes 217
Iron and Steel, Determination of Phosphorus in, Ammonium Phosphomolybdate in the G. Chesneau 53
—— Chemical Analysis of A. A. Blair (Review) 251
—— Corrosion of Heyn & Bauer ... 212
—— Corrosion of Dr. W. H. Walker ... 435
—— in Lead Slag 99
—— Ore and Coal Resources, Conservation of A. Carnegie 248
—— —— Cement in Sea Water 178
—— —— from Natal 63
—— Pit Props, Elastic 218
—— Reducer of, in Volumetric Analysis, Titanous Sulphate as a H. D. Newton 52
—— Rusting of Heyn & Bauer 366
—— Rusting of W. A. Tilden 128
—— Sulphur and Phosphorus J. E. Stead 217

J

Jacobson, C. L., and Coehn, A.—Passivity of Gold 32
Jamieson, G. S., Levy, L. H., and Wells, H. L.—Solubility of Potassium Chloroplatinate 100
—— —— —— Volumetric Estimation of Copper as Cuprous Thiocyanate by Potassium Iodate 54
Jane. W. H.—A Few Notes on Stamp Milling Discussion 21, 46
Do. Reply to 47
Johnson, E. H.—On War Memorial... ... 422
—— J. H.—On Notes on Precipitation ... 351
—— J. P.—The Ore Deposits of South Africa (Review) 322
—— *Tom—Notes on Rand Mining Discussion* 13, 48, 82
Do. Reply to 83
—— *White Labour in Mining* 224
Discussion 305, 389
—— On Rescue Appliances 21
—— On Theory of Blasting with High Explosives 232
Johnston, A. McA.—Gold Crystals 182
—— On Battery and Cyanide Gold Smelting 122
—— On Effect on Amalgamation, &c. ... 429
—— On Gases Resulting from the Use of High Explosives 151
—— On Gold Law Regulations 290
—— On Precipitation of Gold from Solution by Carbonaceous Matter ... 328, 329

PAGE

Johnston, A. McA.—On Routine Assaying on a Westralian Mine 187
—— On South African Coals and Their Economics 333, 391
—— On Zinc Dust Precipitation 76
—— Returning Thanks for Election as President 425
—— Vote of Thanks to President 6
Jones, B. F.—Dust in Coal Mining 102
—— J. A.—On Zinc Dust Precipitation ... 76
Jordis and Rosenhaupt—Oxidation of Metals 57
Jupiter Mine 216

K

Kahn, M.—Reinforced Concrete 178
Kalgoorlie Goldfield, Metallurgy of the G. W. Williams 31
Keane, Dr. C. A., and Burrows, Dr. H.—Automatic Estimation of CO_2 in Furnace Gases... 54
Kendall, Jr., G. D., and Ekeley, J. B.—Determination of Tungstic Acid 98
Kennedy, J.—On a Few Notes on Stamp Milling 46
Kershaw, J. B. C.—Electro-Metallurgy ... 102
Kestner, H.—Rescue Appliances: Lessons from Glencoe Discussion 21, 41
Reply to Discussion 42
Kieselguhr, The Uses of 138
Kilby, J. W. (Obituary) 167
Kingston, C. B.—New Type of Native Compound Building of all Metallic Construction—Discussion 22
Do. Reply to ... 81
—— On Coloured Wrappers for Explosives ... 240
—— On Gases resulting from the use of High Explosives 152
—— On Routine Assaying on a Westralian Mine 187
—— On Small Mines of Rhodesia 166
Kirkland, F. S.—To prevent Crystallisation of Shafts 134
Knights Central Mine 216
Knox, H. H.—On Certain Errors in Computing Ore Values... 135
Koomans—Determination of Tin in Tinstone 30
Kotzé, R. N.—At Annual Dinner 353
—— On Coloured Wrappers for Explosives . . 291
—— On Gold Law Regulations 290

L

Laboratories, Small Chemical, Design and Equipment of R. K. Meade (Review) 103
Laboratory, Acetylene in the 169
—— *Comparison of Tube Mill Pebbles Prof. G. H. Stanley—Discussion* ... 120
Do. Reply to 162
Labour Chart, Mine J. Macdonald ... 444
—— *Efficiency, Westralian Wet Crushing Plants, with some notes on G. W. Williams—Discussion* 24
Do Reply to ... 270
—— *White, in Mining Tom Johnson* ... 224
Discussion 305, 389
Lammon, F. P., and others—Electro Metallurgical Treatment of Copper Slimes... 30
Lance, W. F.—Welcoming Society to Transvaal University College 254
Laschinger, E. J.—On Annual Report ... 422
—— On Effect on Amalgamation, &c. ... 429
—— On S.A. Coals and their Economics ... 432

PAGE
Laschinger, E. J.—On Vote of Thanks to Retiring Council ... 425
Laszczynski Copper Process ... 62
Launay, L. de—The World's Gold (Review) . 371
Launders and Pipes in Reduction Plants, Table of Grades for *C. O. Schmitt* ... 242
—— Carrying Capacity of W. C. Browning ... 441
—— Glass ... 440
—— Linoleum bottomed ... 440
—— Wood ... 440
Law, Gold, Regulations ... 289
Lead Acetate in Cyanidation C. M. Eye .. 368
Lead and Silver in Ores, Analysis of Dr. J. Lövy; and Erratum ... 169, 309
Lead in Ores W. Low ... 213
—— Pig, Analysis of E. W. Buskett ... 129
—— Slag, Analysis of E. W. Buskett ... 99
Leading Nation in Metallurgy C. M. Schwab ... 368
Leather Belts, Steel Bands for ... 218
Lee, C. W.—On Battery and Cyanide Gold Smelting ... 9
Lee's Safety Appliance for Cages ... 101
Lemon, R. D., and Holmes, R. P.—Automatic Filter Press ... 244
Leslie, C. D.—On Theory of Blasting with High Explosives ... 193
Leslie, H. M.—Slimes Filter ... 100
Levy, L. H., Wells, H. L., and Jamieson, G. S.—Solubility of Potassium Chloroplatinate ... 100
—— —— —— Volumetric Estimation of Copper as Cuprous Thiocyanate by Potassium Iodate ... 54
Lewes, V. B.—Carbonisation at Low Temperatures ... 29
Lime in Lead Slag... ... 99
Lime, Quick-, The Influence of Moist Air on *James Gray* ... 396
Liners, Life of Tube Mill ... 241
Linton, Robert—Zinc Dust Precipitation at Cerro Prieto ... 74
Discussion ... 165, 207, 232
—— On Silver Ore Treatment in Mexico 51, 307
Liquid Ammonia F. W. Frerichs ... 367
Litharge as a Substitute for Lead Acetate ... 368
Lloyd, T. H.—The Control of Disinfectants ... 310
Local Sections ... 209
Locomotives in Mines, Petrol ... 447
Lodge, Sir Oliver—Structure of the Atom . 29
Lövy, Dr. J.—Analysis of Lead and Silver in Ores ... 169
Erratum ... 309
Longmuir, P.—Dry Air Blast in Steel Making 217
Louis, H.—Safety Sinking Hooks ... 136
Low, A. H.—Technical Methods of Ore Analysis (Review) ... 66
Low, W.—Lead in Ores ... 213
Lubricating Ropes and Guides ... 318
Lunge, G.—Technical Chemists' Handbook (Review) ... 249
—— Technical Methods of Chemical Analysis (Review) ... 249

M

MacArthur, J. S.—On Reminiscences of the Early Rand ... 204
MacDonald, B.—On Some Features of Silver Ore Treatment in Mexico ... 97
Macdonald, J.—Mine Labour Chart ... 444
Maclaren, J. M.—Gold: Its Geological Occurrence and Geographical Distribution (Review) ... 450
Macquisten Process of Flotation A. Selwyn-Brown ... 411
Magnesia, Determination of ... 99
Magnesium, Separation of Alkalies from Browning and Drushel ... 30
Main Reef West Mine ... 216
Manganese, Effect of ... 51
Mann, E. A.—Detection of Mercury in Nitro-Glycerine Compounds ... 214
—— Fuse Test ... 175
Manual of Practical Assaying H. van F. Furman (Review) ... 450
Manufacture of Cyanide.—P. de Wilde ... 239
Mar, A. del.—Cushioning Vibrations of Cam-Shafts ... 411
Marie, C.—Oxidisability of Platinum ... 54
Martin, H. Stuart.—On Theory of Blasting with High Explosives ... 202
—— On Coloured Wrappers for Explosives ... 240
Matter, Properties of Prof. Sir J. J. Thomson ... 449
Maunsel, R. E. L.—Tool Steel ... 314
Mayer, L. W.—Mining Methods in Europe (Review) ... 451
McCay, L. W.—Action of Hydrogen Sulphide on Alkaline Solutions of Zinc 98, 128
McLeish, J.—Nickel ... 173
Meade, R. K.—Design and Equipment of Small Chemical Laboratories ... 103
Measurements, Stope *O. S. Tonnesen* ... 375
Discussion ... 434
Mechanical Appliances of the Chemical and Metallurgical Industries Dr. O. Nagel (Review) ... 414
Meeting, Special General, Proceedings at December 19, 1908 ... 209
April 17, 1909 ... 351
Mein, W. W.—On Coloured Wrapper for Explosives ... 240
Melrill, G.—A few Notes on Cupelling Gold-Lead Bullion ... 157
Discussion ... 158
Do. Reply to ... 345
—— On Adair-Usher Process .. 50, 96
Members, Election of 1, 37, 69, 105, 141, 181, 221, 254, 289, 325, 373, 417
Membership, Tables of ... 419
Members of Council, Corresponding... ... 2
Mennell, F. P.—Rhodesian Miner's Handbook (Review) ... 451
Mercury in Nitro-Glycerine Compounds, Detection of E. A. Mann ... 214
—— Purification of W. Bettel ... 214
Merits and Demerits of Air Hammer Drills G. E. Wolcott ... 35
Merrill Process at the Homestake Slime Plant, Operation, Costs of... ... 215
Merrin, A. H.—Mine Ventilation ... 245
—— Use of Explosives ... 101
Metallic Alloys G. H. Gulliver (Review)... 323
—— Reduction Process ... 244
Metallurgical Abstracts 30, 55, 100, 130, 171, 214, 244, 278, 312, 368, 411, 437
—— Conditions in Rhodesia ... 80
—— Trials Committee ... 5, 37
Metallurgy of Common Metals L. S. Austin (Review) ... 219

PAGE
Metallurgy of the Kalgoorlie Goldfields G. W. Williams 31
Metals, Boiling Points of Dr. O. P. Watts 100
—— Oxidation of Jordis and Rosenhaupt 57
—— Polishing, for the Microscope M. Robin 244
—— Rare, Production of the W. Giessen 245
Methods of Testing the Efficiency of Ventilation Richards, Wade, Gilbert, Hansen and Talbot 311
Meyer, Hans.—On Effect on Amalgamation, &c. 429
Mexican Institute of Mining and Metallurgy 424
Mexico, Some Features of Silver Ore Treatment in W. A. Caldecott
Reply to Discussion .. . 10
Correspondence 51, 97, 307
Micrographical Study of Cement 321
Microscope, Polishing Metals for the M. Robin 244
Microscopic Examination of Roasted Sulpho-Telluride Ores W. H. Davis ... 62
Middleton Colliery 218
Mill, Tube, Pebbles, A Laboratory Comparison of Prof. G. H. Stanley Discussion 120
Reply to do. ... 162
Mill Pulp Table C. O. Schmitt 208
Mills, Concrete Piles in 98, 128
——— The Rand's New 216
Milling Costs K. S. Handy 368
Milling, Stamp, A Few Notes on W. H. Jane
Discussion 21, 46
Do. Reply to 47
—— Tube, Classification for W. A. Caldecott 312
—— —— Economics of H. W. Fox .. 57
Mills, G. C.—Special Tool Steels 30
Mine Air, Test for Purity of Dr. A. Smith 311
—— Disasters 254
—— Explosions C. B. Ross 247
—— —— Coal Dust as a Factor in H. H. Payne 174
—— Labour Chart J. Macdonald ... 444
—— Pumps, Electrically Driven S. F. Walker 412
—— Tailings on the Rand H. Ingle ... 177
—— Timbers, Eucalyptus for L. W. Symmes 316
—— —— Preservation E. Henry ... 63
—— Ventilation, Experiment in 174
—— —— A. H. Merrin... 245
Mineral? What is a Prof. Gregory ... 414
Minerals in Gold Ores, Unsuspected W. B. Headden 34
—— Occurring in Boiler Waters 408
Mines, Coal Dust in W. E. Garforth .. 246
—— Petrol Locomotives in... 447
—— Trials Committee 37, 69, 254
Mining Abstracts 34, 63, 101, 135, 173, 218, 245, 280, 315, 369, 412, 444
—— Accidents in the United States ... 246
—— and Metallurgical Society of America ... 66
—— Conditions in Rhodesia 80
—— Deep 64
—— Methods in Europe L. W. Mayer (Review) 451
—— —— *The Pilgrims Rest Gold Fields and J. Moyle-Phillips* 293
Discussion 300, 349, 395
—— *Rand, Notes on Tom Johnson*
Discussion 13, 48, 82
Do. Reply to 83
—— Underground E. Taylor 446
Mining, White Labour in Tom Johnson ... 224
Discussion 305, 389
Miscellaneous Abstracts 65, 102, 136, 175, 218, 248, 284, 320, 370, 414, 448
Misleading Analyses of Air (a Correction) Dr. J. Moir 208
Modern Practice in Mining R. A. S. Redmayne (New Book) 219
Moir, Dr. J.—Suggestions for a New Atomic Theory 334
Discussion ... .341, 392, 433
—— *Improved Glasses for Testing for Potassium* 292
—— At Annual Dinner 362
—— Misleading Analyses of Air (a Correction) 208
—— On Coloured Explosives 374
—— On Estimation of CO in Mine Gas ... 265
—— On Gases Resulting from the Use of High Explosives 235, 306
—— Thanks to Dinner Committee 373
—— Dr. W. I.—(Obituary) 98
Moisture and the Strength of Wood... ... 102
Molybdenum 133, 245
—— and its Ores, Commercial Value of ... 171
—— and Vanadium, Separation of G. Edgar 52
—— Extraction and Uses of 171
—— Properties of 171
Moore Clancy Process 438
Morin, L.—Shaft Sinking in Wet Strata ... 135
Mortar Box Pile, Repairing a A. Richardson 24
Mosher, D.—Cyanidation of Silver Ores ... 437
Motors for Mine Pumps 412
Mount Morgan Inquiry Report 315
Mumford, F. T.—On Silver Coating of Amalgamating Plates 222
Muntz Metal Plates 215
—— Sir G. A.—Practice and Science .. 370
Murray, Thornton—The Prevention of Rust... 173
Mursell, Dr. H. Temple—At Annual Dinner 364

N

Nagel, Dr. O.—Mechanical Appliances of the Chemical and Metallurgical Industries (Review) 414
Natal, Iron Ore from 63
Native Compound Building, New Type of, of all Metallic Construction C. B. Kingston—Discussion 22
Do. Reply to ... 81
Neal, J.—Coal Dust Ignition 218
—— W.—Treatment of the Gold and Silver Precipitates at Dos Estrellas ... 443
Neu Essen Colliery 218
New Books, Reviews and 35, 66, 102, 139, 179, 219, 249, 286, 322, 371, 414, 450
—— Form of Chromium 29
—— High Speed Steel—Novo Superior ... 412
—— Method of Making Hydrogen M. M. Beaupre 243
—— *Type of Native Compound Building of all Metallic Construction C. B. Kingston—Discussion* 22
Do. Reply to 81
—— Use for Plaster of Paris 56
—— Zealand, All-Sliming Process in ... 217
Newton, H. D.—Titanous Sulphate as a Reducer of Iron in Volumetric Analysis 52
Nichols, H. G.—Settling Slimes in Cyanide Treatment 56
Nickel J. McLeish 173

PAGE

Nickel and Cobalt, Detection of, when present together H. Grossmann and W. Heilbron ... 129
—— and Copper, Electrolytic Separation of A. Thiel ... 53
—— Determination of ... 100
Nitrogen, Atmospheric, Production of Ammonia from, by means of Peat H. C. Woltereck ... 243
Nitro-Glycerine Compounds, Detection of Mercury in E. A. Mann ... 214
Non-expansive Alloy (Invar) ... 314
Non-Spinning Ropes ... 245
Note on an Experimental Process for Precipitation of Gold from Cyanide Solutions by means of "Decolorising Charcoal" P. de Wilde ... 239
Notes on Precipitation *Mather Smith* ... 300
Discussion ... 351
—— *on a Problem during Shaft Sinking* *C. B. Saner* ... 70
Reply to Discussion ... 300
—— *on Rand Mining* *Tom Johnson*
Discussion ... 13, 48, 82
Do. *Reply to* ... 83
Nystrom, E.—Peat and Lignite, their Manufacture and Uses in Europe (Review) 103

O

Obituary ... 98, 167, 268, 365
Offices, New ... 421
Oil Engines for Small Batteries W. F. Smith ... 32
—— Fuel ... 444
—— *Fuel, The Smelting of Gold Precipitates and Bullion with* *A. Yates* ... 429
On Certain Errors in Computing Ore Values H. H. Knox ... 135
Operation Costs of the Merrill Process at the Homestake Slimes Plant ... 215
Order of Drop in Stamp Mills ... 21, 46, 47
Ore Crushing ... 62
—— Deposits of South Africa J. P. Johnson (Review) ... 322
—— in Bunches or Patches ... 136
—— In Regular Shoots ... 135
—— Iron, from Natal ... 63
—— *Treatment at the Giant Mine, Rhodesia, Description of* *R. C. H. Cooke* ... 152
—— *Treatment, Some Features of Silver, in Mexico* *W. A. Caldecott*
Reply to Discussion ... 10
Correspondence ... 51, 97, 307
—— Values, On Certain Errors in Computing H. H. Knox ... 135
Ores Analysis of Lead and Silver in Dr. J. Lövy: and Erratum ... 169, 309
—— Arsenic in, A Rapid Method for the Estimation of H. E. Hooper ... 26
—— Direct Extraction of Copper from its, by Electrical Methods E. Spargo ... 216
—— Distribution of ... 135
—— Gold, Unsuspected Minerals in W. P. Headden ... 34
—— Lead in W. Low ... 213
—— Silver, Cyanidation of D. Mosher ... 437
—— Silver-Cobalt, Sampling of A. A. Cole 172
—— Sulpho-Telluride, Microscopic Examination of Roasted W. H. Davis ... 62
—— Telluride, Assay of G. T. Holloway and L. E. B. Pearse ... 28

PAGE

Organic Chemistry Dr. J. B. Cohen (Review) ... 35
Origin of the Diamond ... 367
Osdel, E. B. van.—Titration of Zinc in Alkaline Solution ... 28
Oxidation of Amorphous Carbon, Bacteria as Agent in the M. C. Potter ... 138
—— of Metals Jordis and Rosenhaupt ... 57
Oxidisability of Platinum C. Marie ... 54
Oxidising Agents, Table Showing Costs of ... 437
Oxygen in Water, Colorimetric Determination of Dissolved ... 311

P

Paint with a new use ... 175
Palladium and Gold in Solution, Estimation of J. Dovan ... 171
Papers submitted during the Year, List of ... 419
Paris, Plaster of, New use for ... 56
Parry, L.—Analysis of Ashes and Alloys (Review) ... 67
Parting and Weighing ... 185
—— *Apparatus, a Cheap Platinum* *Prof. G. H. Stanley* ... 256
Discussion ... 257
Passing of the Tailings Wheel ... 134
Passivity of Gold A. Coehn and C. L. Jacobson ... 32
Patent Applications, Selected Transvaal, 36, 67, 103, 139, 180, 220, 251, 288, 324, 372, 415, 451
Patio Process *C. Perez Duarte* ... 105
Discussion ... 110
Payne, H. G.—The Filtration of Slimes ... 172
—— H. H.—Coal Dust as a factor in Mine Explosions ... 174
Pearce, S. H.—Assay of Slimes Residues ... 401
—— On Adair-Usher Process ... 23
—— On Reminiscences of the Early Rand ... 272
—— On Zinc Dust Precipitation ... 165
Pearse, L. E. B., and Holloway, G. T.—The Assay of Telluride Ores ... 28
Peat and Lignite, their Manufacture and Uses in Europe E. Nystrom (Review) ... 103
—— Production of Ammonia from Atmospheric Nitrogen by means of H. C. Woltereck ... 243
Pebbles, Tube Mill, a Laboratory Comparison of *Prof. G. H. Stanley—Discussion* 120
Reply to *Do.* 162
Peele, Prof. R.—Compressed Air Plant for Mines (Review) ... 219
Pegging in Rhodesia ... 79
Peirce, A. W. K. (Obituary) ... 365
Petrol Locomotives in Mines ... 447
—— Storage of ... 218
Phillips, J. Moyle.—The Pilgrims Rest Gold Fields and Mining Methods ... 293
Discussion ... 300, 349, 395
—— On Notes on Rand Mining ... 13, 82
—— On Small Mines of Rhodesia ... 126
—— On Theory of Blasting with High Explosives ... 203
Phosphate Deposits of Algeria and Tunis ... 64
Phosphomolybdate, Ammonium, in the Determination of Phosphorus in Iron and Steel G. Chesneau ... 53
Phosphorus, Iron, Sulphur and J. E. Stead 217
Piles in Mills, Concrete ... 98, 128
Pilgrims Rest Gold Fields and Mining Methods *J. Moyle Phillips* ... 293
Discussion ... 300, 349, 395

PAGE
Pipes, Capacity of... 320
Pirsson L. V.—Rocks and Rock Minerals (Review) 139
Pit Props, Elastic Iron 218
Pitted Gold Bullion Bars 216
Plaster of Paris, new use for 56
Plate Absorption, Amalgamated Copper E. Halse and W. E. Thomae 214
Plates, Amalgamating, The Silver Coating of W. A. Caldecott 142, 222
—— Copper, Accumulation and Absorption of Gold Amalgam on E. Halse .. 62
—— *Dressings of, Effect on Amalgamation of Different Intervals of Time between the G. O. Smart* 425
Discussion 427
Platinum Amalgam 31
—— Auriferous, Separation of Gold from E. Schardin 56
—— Metals in Chemicals used for Assaying, The existence of J. G. Rose ... 168
—— Oxidisability of C. Marie 54
—— *Parting Apparatus, A Cheap Prof. G. H. Stanley* 256
Discussion 257
Poisoning, Arseniated Hydrogen 219
Polishing Metals for the Microscope Mr. Robin 244
Pollard, W. P.—A Rapid Method of Qualitative Analysis 242
Portland Metallurgical Society, Researches upon the Telluride Gold Ores of Cripple Creek, Colorado 398
Discussion 434
—— Mill (Colorado, U.S.A.) Cyanide Precipitation and Clean-up at the J. M. Tippett 31
—— Process, The 438
Potassium Chloroplatinate, Solubility of G. S. Jameson, L. H. Levy, H. L. Wells 100
Potassium, Improved Glasses for Testing for Dr. J. Moir 292
—— Iodate, Volumetric Estimation of Copper as Cuprous Thiocyanate by, G. S. Jamieson, L. H. Levy and H. L. Wells 54
Potter, M. C.—Bacteria as Agent in the Oxidation of Amorphous Carbon ... 138
Power Consumption in Tube Milling .. 57
Pozzi-Escot, E.—Determination of Succinic Acid in Presence of Fixed Acids in Wines and Fermented Liquids ... 243
Practice and Science Sir G. A. Muntz ... 370
Precipitates and Bullion, Gold, The Smelting of with Oil Fuel A. Yates .. 429
—— at Dos Estrellas, Treatment of the Gold and Silver W. Neal 443
—— Briquetting 443
Precipitation, Cyanide, and Clean-Up at the Portland Mill J. M. Tippett ... 31
—— *from Cyanide Solutions by Zinc Sharings and Dust, A Comparison of Results and Cost A. J. Clark* 222
—— *Notes on Mather Smith* 300
Discussion 351
—— of Gold from Cyanide Solutions by means of "Decolorising Charcoal." P. de Wilde 239
—— —— —— *Solution by Carbonaceous Matter W. A. Caldecott* ...327, 374, 400
—— *Zinc Dust, at Cerro Prieto Robt. Linton* 74
Discussion 165, 207, 232

PAGE
Premier Diamond Mine, Visit to 209
President (R. G. Bevington), The, Remarks by 2, 6, 9, 13, 21, 24, 37, 38, 42, 45, 50, 51, 69, 73, 75, 76, 81, 94, 97, 105, 110, 118, 123, 141, 143, 156, 158, 163, 181, 183, 191, 193, 196, 202, 203, 207, 209, 221, 222, 238, 239, 253, 254, 255, 289, 290, 291, 292, 293, 300, 305, 306, 326, 327, 328, 329, 333, 341, 350, 352, 353, 354, 365, 373, 374, 388, 391, 417, 418, 422, 424, 425, 427, 429, 432
President's (R. G. Bevington) Inaugural Address 2
Press Filter, Automatic R. D. Lemon and R. P. Holmes 244
Prevention of Coal Dust Explosions M. Dautriche 173
—— of Rust Thornton-Murray 173
Price, Sir Thomas.—At Annual Dinner ... 363
—— W. S. V.—On Coloured Explosives ... 374
—— —— On Stope Measurements 389
—— —— White Labour in Mining... ... 389
—— Griffiths, W. P.—on Pilgrims Rest Gold Fields and Mining Methods ... 395
Prisk, T. H.—A Safety Hook for Shaft Sinking 246
Process, Adair-Usher A. Adair
Discussion 23, 48, 94, 118
Do. Reply to 158
—— Cyanide, Historical Summary of the P. Argall 55
—— The Laszczynski Copper 62
—— *The Patio C. Perez Duarte* ... 105
Discussion 110
—— Sintering, Dwight and Lloyd's .. 32
Proceedings at Ordinary General Meeting—
July 18, 1908 1
Aug. 15, 1908 37
Sept. 19, 1908 69
Oct. 17, 1908 105
Nov. 21, 1908 141
Dec. 19, 1908 181
Jan. 16, 1909 221
Feb. 20, 1909 253
Mar. 20, 1909 289
April 17, 1909 325
May 15, 1909 373
(Annual) June 19, 1909 417
—— at Special General Meeting Dec. 19, 1908 209
April 17, 1909 351
Production of Ammonia from Atmospheric Nitrogen by means of Peat H. C. Woltereck 243
—— of Hydrazine from Inorganic Sources — Raschig... 54
—— of the Rare Metals W. Giessen ... 245
Progress upon Witwatersrand 2
Properties of Matter, The Prof. Sir J. J. Thomson 449
Props, Tapered Timber P. Horan ... 369
Protective Coatings for Cyanide Vats W. H. Gaze 61
Pulp, Mill, Table C. O. Schmitt 208
Pumps, Mine, Electrically Driven S. F. Walker 412
Purchase of Coal under Government and Commercial Specifications on the basis of its Heating Value. D. T. Randall (Review) 67
Purification of Mercury W. Bettel ... 214

PAGE

Q

Qualitative Analysis, A Rapid Method of W. P. Pollard 242
Qualities of Alloys 444
Quantitative Estimation of Dust in Air Prof. M. Hahn 34
Quayle, T. W., and others—Electro Metallurgical Treatment of Copper Slimes ... 30
Queries and Replies 98, 128
Quicklime, The Influence of Moist Air on James Gray... 396

R

Rails, Steel Dr. H. Fay 171
Rand, The Distribution of the Gold produced on the Dr. T. Kirke Rose... ... 265
Discussion.. .. ... 349, 432
—— *Mining, Notes on Tom Johnson*
Discussion 13, 48, 82
Do. Reply to 83
—— Regiments Monument 421
—— *Reminiscences of the Early M. H. Coombe* 38
Discussion 97, 123, 204, 227, 272
Do. Reply to 273
Randall, D. T.—Burning of Coal without Smoke in Boiler Plants (Review) ... 67
—— Purchase of Coal under Government and Commercial Specifications on the basis of its Heating Value (Review) ... 67
Rand's New Mills 216
Rapid Method for the Estimation of Arsenic in Ores H. E. Hooper 26
—— Method of Qualitative Analysis, A W. P. Pollard 242
Rare Metals, Production of the W. Giessen 245
Raschig—Production of Hydrazine from Inorganic Sources 54
Re-Capping Winding Ropes G. H. Winstanley 63
Record Speed in Drilling 246
Rectangular Shafts 447
Redmayne, R. A. S.—Modern Practice in Mining (new book) 219
Reduction Plants, Table of Grades for Launders and Pipes in C. O. Schmitt ... 242
Regulations, Gold Law 289
Reinforced Concrete M. Kahn 178
Reminiscences of the Early Rand M. H. Coombe 38
Discussion 97, 123, 204, 227, 272
Do. Reply to 273
Repairing a Mortar Box Pile A. Richardson 25
Replies, Queries and 98, 128
Report, Annual, of Council 418
—— Book for Mining Engineers A. G. Charlton (Review) 103
—— of the International Committee on Atomic Weights, 1909 F. W. Clarke, W. Ostwald, T. E. Thorpe. G. Urbain ... 310
—— on the Mining and Metallurgical Industries of Canada (Review) 415
Rescue Apparatus 35
—— *Appliances; Lessons from Glencoe H. Kestner—Discussion* ... 21, 41
Do Reply to ... 42
Researches upon the Telluride Gold Ores of Cripple Creek, Colorado. The Portland Metallurgical Society, edited by T. B. Crowe 398
Discussion 434

PAGE

Reumaux, M.—Dust Collector for Rock Drills 64
Revenue Account 423
Reviews and New Books, 35, 66. 102, 139, 179, 219, 249, 286. 322, 372, 414, 450
Rhodesia, Small Mines of B. I. Collings ... 76
Discussion 126, 166, 206, 275
Rhodesian Miner's Handbook F. P. Mennell (Review) 451
Rheinpreussen Colliery 218
Rhodium Crucibles, Use of Iridium and Sir W. Crookes 217
Richards, Prof.—Volatilisation of Silver and Gold 33
Richardson, A.—Repairing a Mortar Box Pile 24
—— On Notes on Rand Mining 18
—— On Pilgrims Rest Gold Fields and Mining Methods 349
—— On Stope Measurements 388
Rickard, T. A.—A Guide to Technical Writing (Review) ' 250
Ringe and Fischer--Extraction of Argon ... 130
Rise of Earth Temperature M. Durnerin... 101
Robin, Mr.—Polishing Metals for the Microscope 244
Rock Drills, Dust Collector for M. Reumaux 64
—— Temperature 245
Rocks and Rock Minerals L. V. Pirsson (Review) 139
Rofe, G. P. W. (Obituary) 365
Rope Drives, Steel Bands for 218
Ropes and Guides, Lubricating 318
—— Non-Spinning 245
—— Re-Capping Winding G. H. Winstanley 63
—— Winding and Conducting, Stresses on J. Hindley and J. Stone 318
—— Wire, Bending Stresses in 318
Rose, J. G.—The Existence of Platinum Metals in Chemicals Used for Assaying 168
—— Dr. T. Kirke.—Alloys of Gold and Tellurium 55
—— —— *The Distribution of the Gold Produced on the Rand ..* 265
Discussion 349, 432
Rosenhaupt and Jordis.—Oxidation of Metals 57
Ross, C. B.—Mine Explosions 247
Routine Assaying on a Westralian Mine W. B. Blyth 184
Discussion 347, 393
Correspondence 435
Roux, P. R.—On Suggestions for a New Atomic Theory 392
Rowland, Fred. (Secretary).—Remarks by 1, 37, 69, 141, 181, 221, 253, 289, 325
Rusden, C. E.—On Adair-Usher Process 49, 95
—— H.—On Cupelling Gold-Lead Bullion ... 158
Rust. The Prevention of Thornton-Murray 173
Rusting of Iron Heyn and Bauer ... 366
—— —— W. A. Tilden... 128

S

Safety Appliances in Winding L. R. Fletcher 101
—— Fuse Experiments 148, 150
—— —— *Testing of by X Rays Jas. Thomas* 183
—— Hook for Shaft Sinking T. H. Prisk 246
—— Sinking Hooks H. Louis 136
Salkinson, A.—Further Notes on the Utilisation of Waste Heat in Slimes Settlement 308
Samples, Mine, and Sampling 184
Sampling of Silver-Cobalt Ores A. A. Cole 172
Sands Collecting Vat, Counterweighted Blind for C. W. Dowsett 326

PAGE
Sands Grading Samples ... 190
Sands Return Contrivance, A Low Fall C. W. Dowsett ... 326
Saner, C. B.—Note on a Problem during Shaft Sinking ... 70
Reply to Discussion ... 300
—— Returning Thanks for Election as Vice-President ... 425
—— On Stope Measurements ... 388
Schardin, E.—Separation of Gold from Auriferous Platinum ... 56
Schmitt, C. O.—Mill Pulp Table ... 208
—— *Table of Grades for Launders and Pipes in Reduction Plants* ... 242
—— On New Type of Native Compound Building ... 22
—— On Notes on Rand Mining ... 15
Scholtz, C.- Deterioration of Coal by Weathering ... 65
Schwab, C. M.—The Leading Nation in Metallurgy ... 368
Science and Practice Sir G. A. Muntz ... 370
Screen, The Callow ... 313
Sea Water, Iron Ore Cement in ... 178
Secretary (Fred. Rowland), Remarks by 1, 37, 69, 141, 181, 221, 253, 289, 325
Sections, Local ... 209
Selection of Carbon for Diamond Drill Bits ... 34
Separation of Alkalies from Magnesium Browning and Drushel ... 30
—— of Gold from Auriferous Platinum E. Schardin ... 56
—— of Mercury from Gold ... 6
—— of Silver Chloride and Iodide. H. Baubigny ... 130
—— of Vanadium and Molybdenum G. Edgar ... 52
Settling Slimes in Cyanide Treatment ... 411
—— —— H. G. Nichols ... 56
Sexton, F. P., and Harker, Dr. J. A., Effect of Pressure on the Boiling Point of Sulphur ... 249
Shaft Sinking, Note on a Problem during C. B. Saner ... 70
Reply to Discussion ... 300
—— —— A Safety Hook for T. H. Prisk 246
—— —— in Wet Strata L. Morin ... 135
Shafts, to Prevent Crystallisation of F. S. Kirkland ... 134
—— Timbering ... 16
—— Cam, Cushioning Vibrations of A. del Mar ... 411
—— Circular and Rectangular ... 446
—— Compound ... 17
Sharwood, W. J.—Conversion Tables for Assay Valuations ... 320
Shots, Blown Out D. Foster ... 319
Silica in Lead Slag ... 99
—— Parting Apparatus Prof. G. H. Stanley ... 417
—— Ware, Fused W. Cullen ... 292
Silundum F. Bölling ... 436
Silver and Gold Precipitates at Dos Estrellas, Treatment of the W. Neal ... 443
—— —— Volatilisation of Prof. Richards 33
—— and Lead in Ores, Analysis of Dr. J. Lövy; and Erratum... 169, 309
—— Chloride and Iodide, Separation of H. Baubigny... 130
—— *Coating of Amalgamating Plates W. A. Caldecott* ... 142, 222
—— Cobalt Ores, Sampling of A. A. Cole 172

PAGE
Silver Cupellation Temperatures Prof. C. H. Fulton ... 435
—— in Gold Bullion E. H. Taylor ... 170
—— Ores, Cyanidation of D. Mosher ... 437
—— —— *Treatment in Mexico, Some Features of W. A. Caldecott—Reply to Discussion* ... 10
Correspondence ... 51, 97, 307
Simmer Deep, Ltd., Visit to 216, 350, 402, 425
Simonis, M.—Chromite for Fire Bricks ... 217
Sinking, Shaft, Note on a Problem during C. B. Saner ... 70
Reply to Discussion ... 300
—— —— a Safety Hook for T. H. Prisk 246
—— —— in Wet Strata L. Morin ... 135
Sintering Process, Dwight & Lloyd's ... 32
Skirrow, Dr. F. W.—Estimation of High Temperatures ... 134
Slag, Cement from Blast Furnace ... 30
—— Lead, Analysis of E. W. Buskett ... 99
Slimes, Copper, Electro-Metallurgical Treatment of D. Hollis and others ... 30
—— Filter H. M. Leslie ... 100
—— Filtration of H. G. Payne ... 172
—— Plant, Homestake, Operation Costs of the Merrill Process at the ... 215
—— Residues, Assay of S. H. Pearce ... 401
—— —— A. Whitby ... 401
—— Settling, in Cyanide Treatment ... 411
—— —— H. G. Nichols ... 56
—— *Treatment, Further Notes on the Utilisation of Waste Heat in A. Salkinson* 308
Sliming, All-, Process in New Zealand .. 217
Small Mine, Equipment of ... 79
—— *Mines of Rhodesia B. I. Collings* ... 76
Discussion ... 126, 166, 206, 275
Smart, G. O.—Effect on Amalgamation of Different Intervals of Time between the Dressings of Plates ... 425
Discussion ... 427
—— On Gold Crystals ... 183
—— On Gold Law Regulations ... 291
Smelting, Battery and Cyanide Gold A. Thomas ... 6
Discussion ... 120, 162
Do. Reply to ... 191
Correspondence ... 208
Addendum ... 418
—— *of Gold Precipitates and Bullion with Oil Fuel A. Yates* ... 429
Smith, Dr. A.—Test for Purity of Mine Air... 311
—— *Mather.—Notes on Precipitation* .. 300
Discussion ... 351
—— W. F.—Oil Engines for Small Batteries 32
Smoke Prevention ... 130
Sodium Peroxide as a reagent ... 242
Solubility of Potassium Chloroplatinate G. S. Jamieson, L. H. Levy and H. L. Wells ... 100
Solution, Gold and Palladium in, Estimation of J. Dovan ... 171
Some Features of Silver Ore Treatment in Mexico W. A. Caldecott
Reply to Discussion ... 10
Correspondence ... 51, 97, 307
South African Coals and their Economics A. J. Andrews ... 330
Discussion ... 391, 432
Sparks from Alloys A. von Welsbach ... 315
Spargo, E.—Direct Extraction of Copper from its Ores by Electrical Methods ... 216
Special Tool Steels G. C. Mills ... 30

PAGE
Spontaneous Combustion of Coal Erdmann and Stolzenberg 65
Stable Calcium Hypochlorite 53
Stadler, H.—On Grading Assays and Grinding Efficiencies... 346
Stamp Battery, Four versus Five 46
—— *Milling, A few notes on* *W. H. Jane*
Discussion 21, 46
Do. *Reply to* 47
Stanley, Prof. G. H.—A Cheap Platinum Parting Apparatus 256
Discussion 257
—— *A Laboratory Comparison of Tube Mill Pebbles* *Discussion* 120
Do. *Reply to* ... 162
—— *Silica Parting Apparatus* 417
—— On Battery and Cyanide Gold Smelting 123, 163, 208
—— On Life of Tube Mill Liners 241
—— On Ore Treatment at the Giant Mine ... 156
Stansfield, Dr. A.—The Electric Furnace, its Evolution, Theory, and Practice (Review) 102
Stead, J. E.—Iron, Sulphur and Phosphorus... 217
Steel and Iron, the Determination of Phosphorus in, Ammonium Phosphomolybdate in G. Chesneau 53
—— Bands for Leather Belts 218
—— Castings, Vanadium in 61
—— Chloride of Barium for Hardening ... 101
—— Guides in Shafts 16
—— Making, Dry Air Blast in P. Longmuir 217
—— New High Speed 412
—— Rails Dr. H. Fay 171
—— Sulphur and Corrosion in G. N. Huntly 437
—— Tool R. E. L. Maunsell 314
Steels, Special Tool G. C. Mills 30
Stokes, Ralph.—On White Labour in Mining 305
Stolzenberg and Erdmann.—Spontaneous Combustion of Coal 65
Stone, J., and Hindley, J.—Stresses on Winding and Conducting Ropes 318
Stope Benches after Blasting 194
Stope Measurements *O. S. Tonnesen* ... 375
Discussion 434
Stoping 13, 198
—— at Pilgrims Rest 297
Storage of Petrol 218
Strength of Wood, Moisture and the ... 102
Stresses in Wire Ropes, Bending 318
—— on Winding and Conducting Ropes J. Hindley and J. Stone 318
Stromeyer, C. E.—Action of Gases on Boiler Plates 320
Structure of the Atom Sir O. Lodge ... 29
Study of Ore Deposits for the Practical Miner Dr. J. P. Wallace (Review) 371
Student Membership, Alteration of Rules for 293, 351
Students, Admission of ... 69, 141, 325, 373
Succinic Acid, Determination of, in Presence of Fixed Acids in Wines and Fermented Liquids E. Pozzi-Escot 243
Sucre Mine, Colombia 214
Suggestions for a New Atomic Theory *Dr. J. Moir* 334
Discussion 341, 392, 433
Sulphate, Titanous, as a Reducer of Iron in Volumetric Analysis H. D. Newton 52
Sulphides and Double Sulphides A. Ditte 98
Sulpho- and Ferro-Cyanides, Determining L. M. Green 213

PAGE
Sulpho-Telluride Ores, Microscopic Examination of Roasted W. H. Davis ... 62
Sulphur and Corrosion in Steel. G. N. Huntley 437
—— and Phosphorus, Iron J. E. Stead ... 217
—— Auto-, Printing Process Bauman ... 217
—— Effect of Pressure on the Boiling Point of Dr. J. A. Harker and F. P. Sexton 249
Sunday Work 65
Symmes, L. W.—Eucalyptus for Mine Timbers 316
Symons, B.—Genesis of Metallic Ores and of the Rocks which enclose them (Review) 179

T

Tables for Assay Valuations, Conversion W. J. Sharwood 320
Table for Mill Pulp *C. O. Schmitt* ... 208
—— *of Grades for Launders and Pipes in Reduction Plants* *C. O. Schmitt* ... 242
—— of Membership 419
Taczak & Hinrichsin.—Coke Analysis .. 412
Tailings, Mine, on the Rand H. Ingle ... 177
—— Wheel, The Passing of the 134
Tamping 115
—— Influence of 146
Tanks, To Close Leaks in 61
Tantalum and Tungsten 218
Tapered Timber Props P. Horan ... 369
Taylor, E.—Underground Mining 446
—— E. H.—Silver in Gold Bullion 170
—— W.—On Small Mines in Rhodesia ... 206
Technical Chemists' Handbook G. Lunge (Review) 249
—— Methods of Chemical Analysis G. Lunge (Review) 249
—— Methods of Ore Analysis A. H. Low (Review) 66
—— Societies Common Room 255
—— Writing, A Guide to T. A. Rickard (Review) 250
Telluride Gold Ores of Cripple Creek, Researches upon the *T. B. Crowe* ... 398
Discussion 434
—— Ores, The Assay of G. T. Holloway and L. E. B. Pearse 28
—— Ores, Sulpho-, Microscopic Examination of Roasted W. H. Davis 62
Tellurium and Gold, Alloys of T. K. Rose 55
—— Detection of 212
—— Ores... 438
—— Test for 129
Temperature, High 56
—— Rise of Earth M. Durnerin... ... 101
Temperatures Carbonisation at Low V. B. Lewes 29
—— Determination of Cupellation C. H. Fulton 28
—— Estimation of High Dr. F. W. Skirrow 134
—— Silver Cupellation Prof. C. H. Fulton 435
Test for Purity of Mine Air Dr. Angus Smith 311
—— for Tellurium... 129
Testing of Safety Fuse by X Rays *Jas. Thomas* 183
—— Reprecipitation by Soluble Sulphides ... 11
Theory and Practice of the Iodometric Determination of Arsenious Acid E. W. Washburn 27

PAGE
Theory of Blasting with High Explosives E. M. Weston ... 110
Discussion ... 193, 232
do Reply to ... 343
Thiel, A.—Electrolytic Separation of Nickel and Copper ... 53
Thiocyanate, Cuprous, Volumetric Estimation of Copper as, by Potassium Iodate G. S. Jamieson, L. H. Levy, H. L. Wells 54
Thomae, W. E. and Halse, E.—Copper Plate Absorption... 214
Thomas, A.—Battery and Cyanide Gold Smelting ... 6
Discussion ... 120, 162
do Reply to ... 191
Correspondence ... 208
Addendum... 418
—— H. Musson.—On Theory of Blasting with High Explosives ... 196
—— *Jas.—Testing of Safety Fuse by X Rays* 183
—— Jas. E.—On Counterweighted Blind for Sands Collecting Vat ... 327
—— On Covers for Extractor Boxes ... 326
—— On Gold Law Regulations ... 290
—— On Precipitation of Gold from Solution by Carbonaceous Matter ... 328
—— Student Membership ... 351
Thomson, Prof. Sir J. J.—Properties of Matter ... 449
Tilden, W. A. - The Rusting of Iron .. 128
Timber Preservation, Mine... ... 63
—— Props, Tapered P. Horan ... 369
Timbering at Pilgrims Rest ... 297
Timbers, Mine, Eucalyptus for L. W. Symmes ... 316
Time, Different Intervals of, between the Dressings of Plates, Effect on Amalgamation of G. O. Smart ... 425
Discussion ... 427
Tin Alloys, Cobalt... ... 312
—— in Tinstone, Determination of Koomans ... 30
Tinstone, Determination of Tin in Koomans ... 30
Tippett, J. M.—Cyanide Precipitation and Clean-Up at the Portland Mill (Colo.) 31
Titanous Sulphate as a Reducer of Iron in Volumetric Analysis H. D. Newton 52
Titration of Zinc in Alkaline Solution E. B. van Osdel ... 28
To Destroy Explosives ... 318
To Prevent Crystallisation of Shafts F. S. Kirkland ... 134
Tonnesen, O. S.—Stope Measurements ... 375
Discussion ... 434
Tool Steel R. E. L. Maunsell ... 314
Transvaal University College ... 4
Treatment of Concentrates carrying Copper B. L. Gardiner ... 130
—— of the Gold and Silver Precipitates at Dos Estrellas W. Neal ... 443
Tube Mill Injector ... 217
Tube Mill Pebbles, A Laboratory Comparison of Prof. G. H. Stanley Discussion 120
Reply to Do. 162
—— —— Size of ... 24
—— Milling ... 3
—— —— Classification for W. A. Caldecott 312
—— Economics of H. W. Fox ... 57
Tungsten and Tantalum ... 218
Tungstic Acid, Determination of J. B. Ekeley and G. D. Kendall, Jr. ... 98

PAGE
Turf Mines Shaft ... 70
Turner, Dr. G. A.—Ankylostomiasis in South Africa ... 175
Twenty Years' Progress in Explosives O. Guttman ... 247

U

Umber, Use of ... 50
Underground Mining E. Taylor... ... 446
—— Traction ... 447
United States, Mining Accidents in the ... 246
Unsuspected Minerals in Gold Ores W. P. Headden ... 34
Use of Explosives A. H. Merrin... ... 101
—— of Iridium and Rhodium Crucibles Sir W. Crookes ... 217
—— *of the Vacuum Pump in the Cyaniding of Sand W. A. Caldecott* ... 240
—— of Kieselguhr ... 138
Usher Process, Adair- A. Adair
Discussion ... 23, 48, 94, 118
Do. Reply to ... 158

V

Vacuum Pump in the Cyaniding of Sand, The Use of the W. A. Caldecott ... 240
Valuations, Assay, Conversion Tables for W. J. Sharwood ... 320
Values, Ore, On Certain Errors in Computing H. H. Knox ... 135
Vanadium and Molybdenum, Separation of G. Edgar ... 52
—— Determination of A. M. Wilson ... 52
—— in Steel Castings ... 61
Vats, Circular, Capacity of, per Foot of Depth W. A. Caldecott ... 127
—— Cyanide, Protective Coatings for W. H. Gaze ... 61
Ventilation, Methods of Testing the Efficiency of Richards, Wade, Gilbert, Hanson and Talbot ... 311
—— Mine A. H. Merrin .. 245
—— —— Experiment in ... 174
—— of Factories Dr. J. S. Haldane ... 136
Visit to Dynamite Factory... ...105, 151, 168
—— to Premier Diamond Mine ... 209
—— to Simmer Deep, Ltd... .. 350, 402, 425
Vogelstruis Consolidated Deep Mine ... 216
Volatilisation of Silver and Gold Prof. Richards ... 33
Volumetric Analysis, Titanous Sulphate as a Reducer of Iron in H. D. Newton 52
—— Estimation of Copper as Cuprous Thiocyanate by Potassium Iodate G. S. Jamieson, L. H. Levy and H. L. Wells 54

W

Waihi Grand Junction Mine ... 217
Walker, S. F.—Electrically Driven Mine Pumps ... 412
—— Dr. W. H.—Corrosion of Iron ... 435
Wallace, Dr. J. P.—A Study of Ore Deposits for the Practical Miner (Review) ... 371
Waltereck, H. C.—Production of Ammonia from Atmospheric Nitrogen by means of Peat ... 243
War Memorial ... 422
Ward, J. P.—On Coloured Wrappers for Explosives... ... 291
—— On Diameter of Explosives Cartridges 291, 292

PAGE
Washburn, E. W.—The Theory and Practice of the Iodometric Determination of Arsenious Acid 27
Water, Boiler Feed, and its Treatment .. 408
—— by Coke, Absorption of 314
—— Colorimetric Determination of Dissolved Oxygen in 311
—— in Boreholes, Determining Depth of C. B. Burdick 64
—— Softeners 410
—— Mine, Acid Reaction of 176
Waters, W.—On Estimation of CO in Mine Gas 265
Watson, F. W.—On Precipitation of Gold from Solution by Carbonaceous matter 400
—— On Suggestions for a New Atomic Theory 341
Watts, Dr. O. P.—Boiling Points of Metals... 100
Weighing and Fluxing 184
Weights, Atomic, Exact Relations among the D. F. Comstock 55
—— —— Report of the International Committee on, 1909 310
—— Conversion Tables 321
Weiskopf, Dr. E. H.—The Estimation of Carbon Monoxide in Mine Gas ... 258
Appendix 306
Wells, H. L., Jamieson, G. S., and Levy, L. H.—Solubility of Potassium Chloroplatinate 100
—— —— —— Volumetric Estimation of Copper as Cuprous Thiocyanate by Potassium Iodate 54
Welsbach, A. von—Sparks from Alloys ... 315
West Rand Consolidated Mine 216
Weston, E. M.—The Theory of Blasting with High Explosives 110
Discussion 193, 232
Do. Reply to 343
Westralian Mine, Routine Assaying on a W. B. Blyth 184
Discussion... ... ... 347, 393
Correspondence 435
—— *Wet-Crushing Plants, with some Notes on Labour efficiency G. W. Williams*
Discussion 24
Do. Reply to 270
What is a Mineral? Prof. Gregory ... 414
Wheel, Tailings, The Passing of the ... 134
Whitby, A.—Assay of Slimes Residues ... 401
—— On Cheap Platinum Parting Apparatus... 258
—— On Student Membership 352
White, H. A.—On Adair-Usher Process ... 48
—— On Attendances of Past Presidents ... 422
—— On Battery and Cyanide Gold Smelting 50
—— [illegible] Bullion ... 158
—— O[illegible] the [illegible]
—— On Effect on Amalgamation of Different Intervals of Time between the Dressings of Plates 429
—— On Gases Resulting from the Use of High Explosives 152
—— On Gold Law Regulations 290
—— On Grading Assays and Grinding Efficiencies 238, 395
—— On Precipitation of Gold from Solution by Carbonaceous Matter ... 328, 329
White, H. A.—On Routine Assaying on a Westralian Mine 187, 393
—— On Westralian Wet-Crushing Plants, with some Notes on Labour Efficiency 24
—— On Zinc Dust Precipitation 76
White Labour in Mining Tom Johnson ... 224
Discussion 305, 389
Whitehouse, J.—On Gases Resulting from the Use of High Explosives 236
Wilde, P. de.—Manufacture of Cyanide ... 239
—— Note on an Experimental Process for Precipitation of Gold from Cyanide Solutions by means of "Decolorising Charcoal" 239
Wilkinson, Prof. J. A.—On Cheap Platinum Parting Apparatus 258
—— On Precipitation of Gold from Solution by Carbonaceous Matter 329
—— On S. A. Coals and their Economics ... 391
—— On Suggestions for a New Atomic Theory 341
Williams, G. W.—Metallurgy of the Kalgoorlie Gold Fields 31
—— *Westralian Wet-Crushing Plants, with some Notes on Labour Efficiency*
Discussion 24
Do. Reply to 270
Wilmoth, L. J.—On Battery and Cyanide Gold Smelting 120
—— On Routine Assaying on a Westralian Mine 347, 435
Wilson, A. M.—Determination of Vanadium 52
—— E. B.—Cyanide Process (Review) ... 179
Winding, Accidents in G. H. Winstanley 63
—— and Conducting Ropes, Stresses on J. Hindley and J. Stone 318
—— Ropes, Re-capping G. H. Winstanley 63
—— Safety Appliances in L. R. Fletcher 101
—— Speed in 16, 18
Winstanley, G. H.—Accidents in Winding ... 63
—— Re-capping Winding Ropes 63
Wolcott. G. E.—Merits and Demerits of Air Hammer Drills 35
Wood and Coal, Ignition Points of 134
—— Fireproofing 248
—— Moisture and the Strength of 102
Work, Sunday 65
World's Gold, The L. de Launay (Review) 371

Y

Yates, Arthur—Grading Assays and Grinding Efficiencies 187
Discussion 238, 346, 395
—— —— *The Smelting of Gold Precipitates and Bullion with Oil Fuel* 429
—— Prof. J.—On Notes on Rand Mining ... 48

Z

Zinc, Alkaline Solutions of, Action of Hydro[illegible] Sulphide on [illegible]
[illegible]
[illegible] 74
[illegible] 165, 207, 232
—— in Alkaline Solution, Titration of E. B. van Osdel 28
—— in Precipitation, Use of an Excess of ... 300
—— in Zinc Dust De Koninck 311
—— *Shavings and Dust, Precipitation from Cyanide Shavings and Dust A. J. Clark* 222

TN
1
S66
v.9

South African Institute of
Mining and Metallurgy
Journal

~~Physical &~~
~~Applied Sci.~~
~~Serials~~

Engineering

PLEASE DO NOT REMOVE
CARDS OR SLIPS FROM THIS POCKET

UNIVERSITY OF TORONTO LIBRARY

ENGIN STORAGE